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# THE TRANSACTIONS

OF THE

# Canadian Institute of Mining and Metallurgy

AND OF THE

# Mining Society of Nova Scotia

# 1922

EDITED BY R. P. D. GRAHAM

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VOLUME XXV

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"The Institute as a body shall not be responsible for the statements and opinions advanced in the papers which may be read or in the discussions which may take place at its meetings."—*By-Laws, Sec. 49.*

❖

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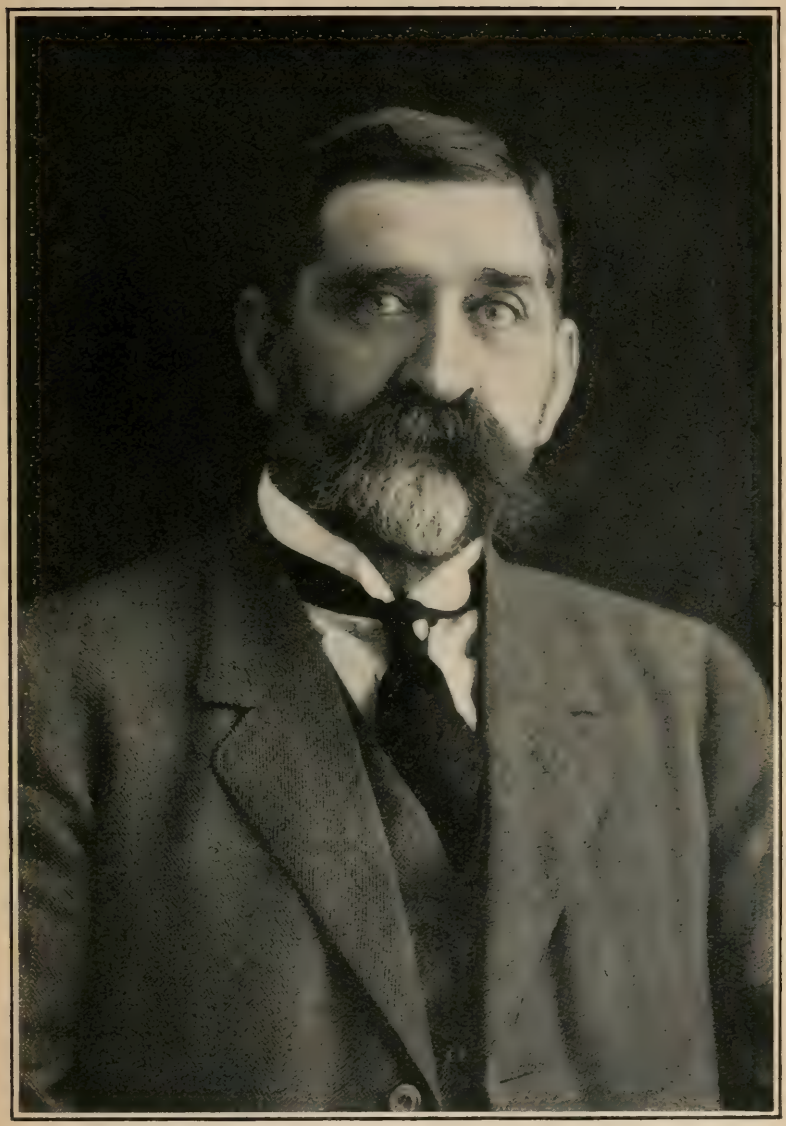

Mr. W. R. WILSON
President of the Institute 1922-1923

# REPORT OF COUNCIL AND FINANCIAL STATEMENT FOR THE YEAR ENDING DECEMBER 31st, 1921.

In submitting to the members of the Institute its report for the year ending December 31st, 1921, the Council desires to express its satisfaction that, notwithstanding the greatly increased expenditure, the affairs of the Society are in a satisfactory condition, and that, while there is an operating deficit for the year, it is smaller than had been anticipated.

The closing months of 1920 witnessed the commencement of a general business depression that continued during the whole of 1921 and was felt throughout the whole world. The allied industries of mining and metallurgy were affected in direct ratio to the depression in the metal markets, which were, for a considerable period of the year, practically non-existent. The depression has been felt most severely in the iron and steel, nickel, copper, lead, and zinc industries. The price of silver made some gains in the latter half of the year, and the costs of gold mining have decreased. These two factors were of particular importance to the precious metal camps of northern Ontario and to some extent relieved the situation occasioned through the curtailment of operations of most of the base metal industries. This general depression in business has, of course, affected the circumstances of many of our members, and while the Institute must face an operating deficit for the year, it is a pleasure to record that our income exceeded expectations, and more particularly that the receipt of membership dues was practically 5% higher than the average for the preceding three years.

Your Council desires to emphasize the immediate necessity for securing increased revenue. The financial obligations of the Institute are increasing. Printing costs for the ensuing year will be approximately $600 higher than for 1921 and $2,000 higher than for 1920, and unless our membership can be increased considerably during the coming year, it will be necessary to draw upon our capital to a further extent.

## SECRETARY-TREASURER'S VISITS TO BRANCHES.

During the year, the Secretary-Treasurer visited over forty different localities and attended twenty branch meetings of the Institute throughout the country, from Nova Scotia to British Columbia. The Branch which formerly had its headquarters at Nelson, B.C., was removed to Rossland, B.C., after consultation with certain Kootenay members, and it is believed that the Branch will enjoy a greater measure of prosperity than it has experienced in the past. The Nanaimo Coal Branch on Vancouver Island was resuscitated and given the new name of the Vancouver Island Branch. The satisfactory results confirm the wisdom of this move. Both of these Branches have since shown decided indications of activity, and it is hoped they will meet with continued prosperity.

Your Council is of the opinion that an annual tour of the branches by the Secretary would be most valuable in strengthening the bonds between headquarters and branches, kindling active interest in the Institute, and thereby increasing membership. There is, unfortunately, no guarantee that this trip can be carried out each year until the annual revenue of the Institute is considerably improved.

## ENGINEERING LEGISLATION, PROVINCE OF ONTARIO.

The question of Engineering Legislation for the province of Ontario was again brought to the attention of the Institute and, at the direction of the President, the Secretary-Treasurer mailed a copy of the proposed Engineers Bill to each Ontario member, together with a ballot paper for the purpose of taking branch votes on this question. As the matter is strictly of a provincial nature, the vote will be taken by the respective Ontario branches, and, when completed, will be published in the BULLETIN for the information of all members.

It is expected that the result of this ballot will be available for publication in the March issue of the BULLETIN.

## DOMINION MINES ACT.

At a Special Meeting of Council held in Edmonton, Alta., on the 16th of September, 1921, a Committee was appointed

to investigate the present Dominion mining regulations and to bring in a report thereon for the next Annual Meeting of the Institute.

## NICKEL COINAGE.

An announcement was made in the public press of last April that the Canadian Government had decided to introduce the minting of five-cent nickel coins. Some delay was at first occasioned in securing the properly annealed blanks for coining purposes, but these difficulties have been overcome and the Royal Mint at Ottawa is now engaged in minting these new coins. The Institute has long advocated the use of nickel coinage in Canada, not only because we possess the most important nickel deposits in the world, so that the metal is peculiarly Canadian, but also because nickel possesses certain features which render it eminently suitable for coins of low value. It is gratifying to record that our efforts have at last borne fruit.

## ROYAL SCHOOL OF MINES FRECHEVILLE RESEARCH FELLOWSHIPS.

There were two enquiries received during the year concerning these fellowships, but no applications have been recorded.

## AFFILIATED STUDENT SOCIETIES.

There are now four Student Societies in affiliation with the Institute. These are:—

The Mining and Geological Society, University of Alberta, with a membership of eleven.

The Mining and Metallurgical Club, University of Toronto, with a membership of thirty-eight.

The Mining and Metallurgical Society of Queen's University, with a membership of twenty-nine.

The Mining and Metallurgical Society of McGill University, with a membership of fifty-six.

The Students' Competition for the President's gold medal and other prizes was revived by last year's Council, after having been in abeyance for several years. It is expected that a number of papers will be entered for this competition.

## AMENDMENT TO BY-LAW NUMBER THIRTY.

The scrutineers appointed to examine the ballot for the proposed amendment to the By-Laws in connection with the creation of an Executive Committee, have reported the amendment carried by more than two-thirds of the votes cast. It will now be necessary for the incoming Council, at their first regular meeting after the next Annual Meeting, to appoint this Executive Committee.

## MEETINGS.

The Twenty-third Annual General Meeting of the Institute was held in the Windsor Hotel, Montreal, on March 2nd, 3rd, and 4th, the attendance being two hundred and forty. Many excellent papers on a variety of subjects were presented. Probably the most valuable session of the meeting was that devoted to papers on recent oil discoveries in the west, supplemented with motion pictures descriptive of the oil industry in all its phases. The guests of honour at the banquet included Sir Arthur Currie, Mr. J. M. R. Fairbairn, Col. A. S. Dwight, Mr. Edwin Ludlow, Dr. H. Foster Bain, and Dr. H. C. Lane.

The Third Annual Western Meeting of the Institute was held in the Macdonald Hotel, Edmonton, on September 14th, 15th, and 16th, the attendance being one hundred and twenty-five. The papers presented dealt chiefly with the subject of coal mining, and brought forth considerable discussion. The guests of honour at the banquet included the Hon. George Handley, the Hon. Charles Stewart, and Mr. Duggan, the Mayor of Edmonton. Following the meeting an excursion took place to what is locally known as the "Coal Branch" of the C.N.R., just east of Jasper. This excursion was participated in by nearly sixty members and friends, who were hospitably entertained by officials of the Cadomin Coal Company,

the Sterling Coal Company, and the Yellowhead Coal Company.

The Twenty-ninth Annual Meeting of the Mining Society of Nova Scotia was held in the Halifax Hotel, Halifax, on the 5th and 6th of April. The programme included an excellent series of papers, mainly relating to various phases of coal-mining, and the sessions throughout were well attended, some fifty members being present.

The Second General Meeting of the British Columbian Division took place in Vancouver on the 9th and 10th of February and was well attended by over two hundred members and guests. A notable feature of this meeting was the absence of technical papers at the various sessions and of speeches at the social functions. Instead of the customary papers, selected topics were introduced for general discussion, in which the members evinced considerable interest. The meeting was a decided success and will without doubt become an annual fixture.

## AFFILIATED SOCIETIES AND BRANCHES.

These now consist of the following organizations:—

| | |
|---|---|
| Nova Scotia | The Mining Society of Nova Scotia. |
| Quebec | Thetford District Branch, |
| | Montreal Branch. |
| Ontario | Ottawa Branch, |
| | Hastings District Branch, |
| | Toronto Branch, |
| | Sudbury Branch, |
| | Cobalt Branch, |
| | Porcupine Branch. |
| Manitoba | Manitoba Branch. |
| Alberta | Northern Alberta Branch, |
| | Rocky Mountain Branch. |
| British Columbia | British Columbian Division, |
| | Rossland Branch, |
| | Vancouver Branch, |
| | North Coast Branch, |
| | Vancouver Island Branch. |

Practically all Branch organizations have been active throughout the year. Naturally some of the Branches have displayed more energy than others, but all have played a considerable part in sustaining the Institute.

## PUBLICATIONS.

Twelve numbers of the MONTHLY BULLETIN have been published during the year, and, while the expenses in connection therewith have considerably increased, it is believed that the service maintained has been appreciated. It is particularly desired to increase the advertising revenue from the BULLETIN, as this would in large measure relieve the financial strain on the Institute's resources. The Council desires to express its appreciation to the special contributors to the BULLETIN, amongst whom are the Hon. Robt. Drummond, Mr. W. R. Rogers, Mr. R. R. Rose, Mr. Robt. Dunn, Mr. R. H. Hague, Mr. R. J. Lee, Mr. R. Strachan, Mr. A. J. C. Nettell, Mr. R. H. Hutchison, Mr. W. J. Woolsey, and others. Thanks are due to the *Engineering and Mining Journal* for continued contribution of the Metal Market Review.

Volume XXIII of the ANNUAL TRANSACTIONS was not issued until late in December, owing to printers' strikes and other troubles. Volume XXIV is now in process of printing and should be ready for distribution in June, 1922. The warmest thanks are due to all who contributed to the TRANSACTIONS.

## LIBRARY.

No books other than the usual annual publications have been bought for the Library. Sets of TRANSACTIONS, comprising those volumes still in print, were presented to the libraries of the Universities of British Columbia and Manitoba, and to the British Columbian Division.

## THE FEDERAL DEPARTMENT OF MINES.

Shortly after the announcement of the result of the Federal elections, in December, 1921, the President addressed the

following telegram to the Premier-elect, and at the same time instructed the Secretary-Treasurer to advise all branches of the Institute to telegraph the Premier-elect endorsing the President's recommendations.

Coniston, Ont., Dec. 20th, 1921.

Hon. W. L. Mackenzie King,
Parliament Buildings, Ottawa, Ontario.

Dear Sir.—

On behalf of the Canadian Institute of Mining and Metallurgy, I tender you sincere congratulations on your accession to the position of Premier, and express the hope and confidence that Canada's greatest industry of the future, namely, Mining, will be represented in the Cabinet you are now forming by the broadest minded, most capable and best informed man to be found, one who clearly realizes that Canada's future greatness in industry rests more on her minerals than on any other natural resource. It is a startling and not generally realized fact that ninety per cent of Canada contains inconceivable wealth in minerals and that not over fifteen per cent is suitable for profitable agriculture. The ultimate permanent material sources of wealth of any country are its soil and its minerals. We hope that in the great pressure of public affairs these vastly important, silent, but incontrovertible facts will receive their proper weight in organizing the new Cabinet.

Yours truly,

THE CANADIAN INSTITUTE OF
MINING AND METALLURGY,

C. V. CORLESS, *President.*

The receipt of these telegrams was acknowledged by the Premier's Secretary, who advised that the Hon. Mackenzie King would give the matter his careful consideration.

## MEMBERSHIP.

The membership roll of the Institute, so far as members are concerned, stands practically the same as at December

## CLASSIFICATION OF MEMBERSHIP AND CHANGES DURING THE YEAR 1921.

| Classification | Totals at Jan. 1st, 1921 | Changes during calendar year 1921 | | | | | | | Totals at Jan. 1st, 1922 |
|---|---|---|---|---|---|---|---|---|---|
| | | Elected and accepted | Elected, acceptance not yet received | Election void | Election to date from Jan. 1st, 1922 | Deaths | Resignations | Removals | |
| Patrons | 2 | .. | .. | .. | .. | .. | .. | .. | 2 |
| Life members | 20 | 4 | .. | .. | .. | .. | .. | .. | 24 |
| Honorary members | 6 | 2 | .. | .. | .. | .. | .. | .. | 8 |
| Members | 1087 | 120 | *15 | *6 | *24 | 8 | 32 | 94 | 1073 |
| Associate members | 268 | 49 | * 9 | *2 | .. | 2 | 11 | 34 | 270 |
| Ex-officio members | 50 | 4 | .. | .. | .. | .. | .. | 5 | 49 |
| Corresponding members | 14 | .. | .. | .. | .. | .. | .. | .. | 14 |
| Student members | 34 | 17 | .. | *1 | .. | .. | .. | 5 | 46 |
| Affiliated student members | 116 | .. | .. | .. | .. | .. | .. | .. | 134 |
| Totals | 1597 | 196 | *24 | *9 | *24 | 10 | 43 | 138 | 1620 |

4 members took out life membership; deduct from members.
2 ex-officio members became honorary members.
3 ex-officio members became members.
1 associate member became a member.
*these are not counted in totals.

31st, 1920. A number of elections took place during the year, but these were offset through the deaths, resignations, and removals of old members.

The changes which occurred throughout the year are set forth in the accompanying tabulated statement, which also compares the membership lists as at December 31st, 1920, with the lists as at December 31st, 1921.

The loss by death of the following members is recorded with deep regret:

F. I. Daniels,
T. M. Daulton,
Alex. Faulds,
M. R. Jennings.
Thos. J. Lloyd,
Robt. McKay,
Alex. Sharp,
H. D. Symmes,
Stuart M. Thorne,
W. P. Williams,

The following resignations were accepted by Council during the year:—

J. M. Clark,
J. C. Haas,
A. E. Hall,
J. D. Hurd,
H. S. Robinson,
T. W. Stiles,
Sir John Cadman,
M. W. Garman,
W. J. Haines,
G. C. McTavish,
J. F. McClelland,
H. J. Fetter,
Dan Lindeborg,
J. A. Baker,
J. H. Warner,
Edw. Bagley,
J. T. Donald,
P. E. Dulieux,
H. Smeaton,
R. A. Stinson,
J. J. Turnbull,
A. F. Turner,
R. H. James,
Robt. Jacobs,
Gilbert Rigg,
A. Fournier,
R. S. G. Stokes,
Jas. White,
F. M. Sylvester,
H. Giegerich.
Louis D. Huntoon,
W. S. Morse,
John Bain,
G. E. Bolton,
L. D. Burling,
Sir John W. Carson,
H. R. Davis,
A. E. Doucet,
W. E. C. Eustis,
E. W. Lamarque,
W. H. Trewartha-James,
W. J. Wilson,
N. Wright,

C. V. CORLESS,
*President.*

G. C. MACKENZIE,
*Secretary-Treasurer.*

## TREASURER'S REPORT.

The current revenue has been insufficient to meet all obligations, including depreciation and reserves; and while there was a cash balance at the end of the year, the amount of this balance is not sufficient to cover outstanding liabilities. The receipts and disbursements during the year ending December 31st, 1921, were as follows:—

### *Receipts*

| | | | |
|---|---|---|---|
| Cash in Bank, Jan. 1st, 1921 | | | $4,629.45 |
| Subscriptions: Annual | $11,096.50 | | |
| Students | 262.00 | | |
| Arrears | 1,154.56 | | |
| Advance | 204.10 | | |
| Life | 400.00 | | |
| | | $ 13,117.16 | |
| Publications: Sales | 345.10 | | |
| Advertisements | 2,885.24 | | |
| Binding | 382.00 | | |
| Index | 10.75 | | |
| | | 3,623.09 | |
| Revenue from Investments | 1,310.00 | | |
| Bank Interest | 270.72 | | |
| | | 1,580.72 | |
| Dominion Government Grant | | 3,000.00 | |
| | | | 21,320.97 |
| | | | $25,950.42 |

### *Disbursements*

| | | | |
|---|---|---|---|
| Publications: | | | |
| Printing and distributing | 4,594.84 | | |
| Editors' salaries | 2,400.00 | | |
| | | 6,994.84 | |
| Miscellaneous: | | | |
| Rent, rates and taxes | 1,675.26 | | |
| Telegrams and telephones | 243.53 | | |
| Office salaries | 8,220.00 | | |
| Postage | 606.85 | | |
| Printing and stationery | 959.84 | | |
| Express | 61.73 | | |
| Travelling expense | 1,999.01 | | |
| Annual Meeting | 499.61 | | |
| Thetford Branch | 25.00 | | |
| General expense | 225.57 | | |
| Grant to the Canadian Engineering Standards' Association | 100.00 | | |
| Buttons | 81.00 | | |
| | | $14,697.40 | |

| | | | |
|---|---|---|---|
| Permanent: | | | |
| Library | $ 19.67 | | |
| Furniture and fixtures | 176.13 | | |
| | | $195.80 | |
| | | | $21,888.04 |
| Cash in Bank, Dec. 31st, 1921 | | | 4,062.38 |
| | | | $ 25,950.42 |

Audited and Verified,
(Signed) **P. S. Ross & Sons,**
Chartered Accountants.
Montreal, January 9th, 1922.

G. C. MACKENZIE,
*Treasurer.*

## REVENUE STATEMENT BY FINANCE COMMITTEE.

### *Revenue*

| | | |
|---|---|---|
| Revenue per Cash Statement | | $ 21,320.97 |
| Add Subscriptions collected in 1920 for 1921 | | 316.00 |
| | | 21,636.97 |
| Less: Life memberships | $ 400.00 | |
| Subscriptions paid in advance | 204.10 | |
| Interest on Investments 1920 received in 1921 | 297.50 | |
| | | 901.60 |
| | | $ 20,735.37 |

### *Expenditure*

| | | |
|---|---|---|
| Disbursements per Cash Statement | $ 21,692.24 | |
| Permanent fixtures | 195.80 | |
| Unpaid accounts at Dec. 31st, 1921 | 4,288.47 | |
| | | 26,176.51 |
| Excess expenditure over revenue for year | | $ 5,441.14 |

*For the Finance Committee,*
By H. Y. RUSSEL,
J. G. ROSS.

Under instructions of Council, the Finance Committee of the Institute submit the following report:—

Jan. 20th, 1922

To the Members of the Canadian Institute of Mining and Metallurgy:

In addition to the usual Auditors' Statement of the finances of the Institute you will notice a statement by the Finance Committee.

This statement gives the actual revenue of the Institute and the expenditures for the year 1921. You will see that the expenditures exceed the revenue by something over $5,000.

Depreciation has not been allowed for for the year, but as we have already depreciated our furniture and fixtures by 50% we are in good shape in this regard.

The reasons for the above condition are as follows:—

After the Annual Meetings of 1920 (in Toronto) and 1921 (in Montreal), the Council decided that to carry out the concensus of opinion expressed at those meetings it was necessary that the Institute take a more active interest in the affairs of the Industry and also get in closer touch with our Branches, get their ideas as to what the Institute should do, keep them informed of what other Branches were doing, and so interest every member of the Institute in what was being done.

To do this it was necessary to have a Secretary of standing, ability, and ideas; it was also necessary to have him footloose, so that he could do the necessary travelling without interfering with the routine work of the Institute.

After consulting the Finance Committee the Council decided to increase the salary of the Secretary, to allow $2000.00 for travelling expenses for 1921, and to appoint salaried Editors to take over the editing of the BULLETIN and the TRANSACTIONS. The above extra expenses account for practically all the excess expenditure.

In deciding to carry out the 1921 programme, both the Council and the Finance Committee realized that such a deficit would result and both strongly believed that the programme was for the best interests of the Institute.

The financial condition of the Institute is good. It has ample reserves to meet the over-expenditure of 1921, and if necessary, further over-expenditures.

The object of this report is to bring out discussion at the Annual Meeting at Ottawa in March, when reports will be placed before the Meeting showing what has been started and what has been accomplished in 1921, and also what is necessary to reduce expenditure for 1922 without seriously restricting the active policy started at the beginning of 1921.

*For the Finance Committee,*
BY H. Y. RUSSEL,
J. G. ROSS.

# PRESIDENTIAL ADDRESS.

By C. V. Corless

Annual General Meeting, Ottawa, March, 1922.

In the telegram [1] on behalf of the Institute sent to the Prime Minister of Canada on his accession to power, attention was called to the relative future importance of the Mining Industry. The surprise, rather than criticism, expressed in press comment on this telegram, led me to think, rightly or wrongly, that a somewhat popular non-technical discussion of the facts and inferences giving rise to this opinion might be of real value to citizens whose education, daily occupation, and surroundings have contributed to form in their mind an inadequate view as to the relative importance of the mineral resources of Canada, and as to the fundamental place which the discovery, mining, and metallurgical preparation of these resources occupies in the progress of civilization.

We live in a country whose greatest permanent material sources of wealth are its soil and its minerals. But few people realize, however, that not more than about fifteen per cent of its entire surface is suitable for profitable cultivation. Of the

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1. "On behalf of the Canadian Institute of Mining and Metallurgy, I tender you sincere congratulations on your accession to the position of Premier, and express the hope and confidence that Canada's greatest industry of the future, namely, Mining, will be represented in the Cabinet you are now forming by the broadest minded, most capable and best informed man to be found, one who clearly realizes that Canada's future greatness in industry rests more on her minerals than on any other natural resource. It is a startling and not generally realized fact that ninety per cent of Canada contains inconceivable wealth in minerals and that not over fifteen per cent is suitable for profitable agriculture. The ultimate permanent material sources of wealth of any country are its soil and its minerals. We hope that in the great pressure of public affairs these vastly important, silent, but incontrovertible facts will receive their proper weight in organizing the new Cabinet."

two largest provinces, generally considered as mainly agricultural, this proportion is probably less than ten per cent. I shall endeavour to show that the remaining eighty-five per cent of Canada, and a large part even of the fifteen per cent, contain mineral treasures of inconceivable value.

Everyone knows, of course, that mineral deposits are generally difficult to discover. Information regarding them is not as easily obtained as in the case of timber or soil. The trails for many years passed over some of the richest native silver veins at Cobalt. This fact and the added fact that, because of this difficulty of discovery and estimate, we must pass from accurate and slowly acquired knowledge to reasonable inference, if we are ever to form any kind of estimate of our mineral heritage, easily account for the general failure to appreciate the importance of Canada's mineral resources. But just as a business corporation, if it fails to make reasonable forecasts, can have no adequate business policy, so will there be no adequate public mining policy, if we as citizens, through our representatives in the Dominion and Provincial Governments, either because of the difficulties alluded to or for other reasons, fail to exercise reasonable foresight with regard to our mineral resources.

Before dealing with this question, I wish to offer quite briefly a reason for regarding mining as of great fundamental importance to modern scientific civilization. It is the basic industry of progress, as agriculture is of subsistence.

Civilized progress, as we understand it to-day, is based on at least four fundamental conditions: justice, security, a spirit of toleration, and growth and spread of knowledge. There may be other factors, but these are fundamental and they are a minimum. If there is any doubt about these being essential, we have only to try to conceive of a society characterized by injustice, insecurity of life or property, a spirit of intolerance, or a static condition of knowledge with spreading ignorance, to become convinced that its decay is already begun.

Now, growth of each of these four essentials to civilized progress, but more particularly increase of knowledge, is conditioned on a reasonable amount of leisure. A life of

drudgery for mere subsistence contributes nothing to civilized advancement. In ancient civilizations, leisure for the thinking classes generally was obtained by a system of slavery. In the palmy days of Athens, she had 20,000 or 30,000 citizens and 400,000 slaves or bondmen. But, though knowledge did increase, such systems were founded on injustice, and this fatal weakness in their foundation made downfall inevitable. If the leisure essential for increase of knowledge is obtained by enslaving our fellow men, one of the supporting pillars of any civilization is smitten with decay and the final collapse of the social structure is certain.

Thus, in these older societies, knowledge of certain kinds increased, and, in consequence, advances to a certain degree were made in each succeeding civilization; but, because of the weakness referred to, civilization after civilization ran its course and met its downfall. Then man slowly discovered that knowledge along certain definite lines, combined with invention, gave him the means of controlling the forces of nature, and of gradually reducing the drudgery which had been his chief incentive in enslaving the less fortunate of his fellows. Since then he has directed his pursuit of knowledge more and more toward increasing his control of these forces. His initial successes in gaining this control mark the beginning of an epoch in human progress quite as important as his discovery of fire. Not only is such success enormously stimulating to his mind, but each forward step adds to his leisure for the further pursuit of knowledge. This reaction, which thus causes acceleration, is the key to the marvellous increase in the rate of material progress attained during the past half century. Every step forward in harnessing nature reduces drudgery and both stimulates and releases the human mind for further conquests. But, as already suggested, knowledge in the abstract is insufficient. It must be applied before control is gained and the human mind thus set free to continue its researches.

This is a convenient point at which to digress briefly in order to make passing reference to the peculiar narrowness of view occasionally shown by pure scientists, a few of whom assume an attitude of smug superiority toward those whose chief occupation consists in seeking practical applications of

the discoveries of science. This attitude, when it exists, shows a failure on the part of those assuming it to appreciate the efforts and achievements of their fellow workers, without which the leisure, and generally even the very materials and appliances, for continuing their own researches would not have existed. Moreover, researches in pure science find their true fruition in practical results, and these fruits will generally be found to contain the seeds from which new knowledge will spring. Both pure and applied science will make more real and rapid progress when they proceed hand in hand. They are not antagonistic, and the spirit of mutual helpfulness should always characterize their devotees. Indeed, successful application is frequently the best, and sometimes the only, test of the genuineness, the completeness, or the limits, of a supposed discovery.

Now, application of knowledge to the control or transformation of nature's materials and energy for useful purposes could make very little progress without the use of metals. Before this stage of civilized progress was reached, very little use for metals was found except for simple tools or for purposes of adornment. The greatest final, though perhaps not immediate, stimulus in the search for new metals, for new properties of the known metals and their alloys, and for new supplies of the metals, has resulted from this desire to increase control over nature, to harness her and cause her to do man's work, while he reduces his drudgery, rises above the level of mere subsistence, and frees his mind for yet further conquests. In so substituting the conquest of nature for the enslavement of part of the human race, thus opening the door of opportunity for progress to a steadily increasing percentage of the population, modern civilization is better fitted to survive than any preceding form. If our increasingly scientific civilization can properly be safeguarded against some new and insidious form of slavery as, for example, slavery to organization or to machinery; if it can adequately correct that threatening form of injustice, the injustice of unfair distribution of its increasing productivity; if it can free itself from insatiable, and therefore enslaving, material wants, often dictated by tyrannous social conventions and stimulated by a press devoted to commercial aims rather than to the spread of knowledge

and truth; if it can find wise use for, and avoid abuse of, the growing leisure made possible by the increasing use of nature's energy; or, to sum these conditions all up, if our modern civilization can be made sane and moral, by fixing in the minds of its youth a truer realization of the relative values of its material and its immaterial products; then the line along which it is developing so rapidly should give it a better chance of survival than any preceding form. But the very strength I have pointed out constitutes its greatest danger, though there are hopeful signs that many men are beginning to see more and more clearly what these dangers are and, in some cases at least, are earnestly considering the best ways to meet them. But for our present line of thought the main facts are clear. The progress of our existing scientific civilization, barring needless abuses, will accelerate. This accelerating progress, on its material side, is based on the use of metals. Hence we seem to have sound and adequate reason for the opinion that the increased development of mining is more essential to the progress of western civilization, and therefore more certain, than that of any other basic industry.

I have given what appear to me to be valid and fundamental reasons for my belief that the demand for minerals per unit of population will not diminish as civilization progresses, but will rather increase, probably beyond what we now conceive, even as this demand during the past century has surpassed the utmost dreams of a century ago. This acceleration appears to be inherent in our scientific progress.

Let us next review a few facts, well-known to most members of this Institute, but probably not so well-known to the average well-informed man whom we meet — facts bearing on Canada's future place as a producer of minerals. We all are pleased with the progress of Mining in this country in the past quarter of a century; yet, compared with its certain future, the mining industry in Canada is scarcely in its infancy Let us briefly summarize the principal facts and inferences substantiating this statement.

For almost two decades, a former President of this Institute, Dr. Willet G. Miller, has been calling attention to the importance of Canada's enormous pre-Cambrian area,

nearly surrounding Hudson Bay, occupying more than half of the entire surface of our country, the greatest single exposure (greater indeed than all others in the world added together) of our mother earth's basement, wherein, as we are recently finding, her richest treasures are stored for future use. Dr. Miller has not been alone in this, but he perhaps more than any other has given his ability, his voice, and pen, without stint and, in his position as Provincial Geologist of Ontario, has directed the work of a brilliant group of geologists, who have greatly assisted the mine operators and prospectors and who, incidentally, have demonstrated the soundness of his views, with regard to a small part of this enormous Archean area. Dr. Wallace has for a shorter period performed a similar service in Manitoba.

Now Canada, as is well known, has a monopoly of this huge area, excepting only a small spur into New York State and a slight projection southwest of Lake Superior, into Michigan, Wisconsin and Minnesota. This projection, though only two or three per cent of the whole pre-Cambrian area, is beyond doubt one of the richest mining districts in the world. It contains the famous Lake Superior iron mines, which produce annually more than eighty per cent of the entire output of iron ore in the United States; which have still almost inexhaustible reserves of ore remaining; and which, more than any other single factor, have made that country pre-eminent in iron and steel production. It contains also the famous copper deposits of Keweenaw Peninsula in Michigan, which have been worked for three-quarters of a century; which have for years, if we except the present period of depression, produced more than two hundred million pounds of copper annually; and which have still as reserves very great but unknown tonnages of copper ore.

On the Canadian side of Lake Superior, real prospecting in this vast pre-Cambrian area is scarcely begun. The Bruce copper mines were easily discovered many years ago, since they were along the shore of Lake Huron. The story of the accidental discovery of nickel ore in the Sudbury District thirty-seven years ago is well known. Though surface prospecting soon located most of the outcrops, some of the

principal orebodies, and the enormous importance of the nickel-copper deposits of this district as a whole, have been demonstrated only by extensive and costly exploration with the diamond drill, chiefly within the past ten or twelve years. The first discovery at Cobalt, during railway construction was as accidental as that at Sudbury: Porcupine, one mine in which may soon reach the position of the world's greatest gold producer, is not far from the same railway. Each of these three camps — Sudbury, Cobalt, and Porcupine — will, during its total life, produce metals of a gross value amounting to hundreds of millions of dollars. Each of these camps has acted as a stimulus to prospecting. Prospectors, generally experienced in them, have found a large number of other nickel-, silver-, and gold-bearing areas, of which some are already reaching steady production; some are being developed only; and others of considerable promise are not yet beyond the stage of preliminary exploration; but all, when considered together, contribute to the impression, which, for those giving the subject close attention, has grown to be a certainty, that these metals, and probably others as well, are very widely and generously distributed.

There have also been found extensive iron ore areas, which require much more detailed exploration than they have yet received. The deposits of iron ore so far discovered present slight, but certainly not insuperable, technical difficulties. Their chief present handicap is competition with the richer and more easily mined Lake Superior ores already mentioned. In this great pre-Cambrian area in Canada, there are already known a number of areas of similar geological age and formation to those associated with the Lake Superior iron ore deposits and undoubtedly others will be found. It is only reasonable to expect that continued geological work, followed by detailed prospecting and exploration, will yet find large and rich iron ore deposits, requiring no concentration, and suitable in composition for steel-making. Probably one or more of these will be found near the shores of Hudson Bay, on both sides of which Keewatin and Huronian areas, similar to those associated with the Lake Superior iron ores, are known. This large inland sea is said to be navigable for about as great a part of the year as Lake Superior. The fact of our great

national need is an added stimulus in the search for accessible bodies of iron ore of higher grade than those already found; and in our attack on the technical problems arising from our already known deposits of iron ore.

Quebec has far the largest share of the great pre-Cambrian shield, which fills ninety per cent, if not more, of the entire province. Cobalt, Porcupine, and some of the newer discoveries already referred to, are near the Quebec boundary. Geological formation and ore deposition bear no relation to interprovincial boundaries established by man. But Quebec has not yet seriously turned her attention to the exploration and geological mapping, much less to the prospecting, of this huge area. The attention of her mining men has so far been largely directed to the rich mineral deposits of the Acadian Highland, which occupies the Eastern Townships and Gaspe Peninsula, and in which asbestos, chrome, copper, lead, zinc, and gold ores, as well as other valuable minerals, have been found. I see no reason to doubt that enormous treasures in minerals in this area await discovery by the thrifty people of this province, whose well-known pioneering ability, which has been so successful with the axe and the canoe, will prove to be equally successful when their keen natural aptitude is specially trained and turned to the search for minerals in this great northland of their province.

In Manitoba, prospecting in the pre-Cambrian area, which occupies two-thirds or more of that province, has barely been begun. Because of lack of transportation facilities and low base metal prices, there is at present but little being done. But diamond drilling is said to have proven twenty million tons of ore at Flin Flon; Mandy mine produced some thousands of tons of ore with so high a copper content as to stand a long sleigh haul and rail transportation to Trail smelter; and numerous prospects, some reported as having much promise, have already been located.

One-fifth of Saskatchewan is occupied by the "Great Shield", and exploration of this area is just beginning. The pre-Cambrian shield also nips a corner of about 2,000 square miles off Alberta.

Northward from the 60th parallel, which forms the northern boundary of the four western provinces, but little

is known of the great pre-Cambrian area, which fills the North West Territory from Hudson Bay almost to Mackenzie river. But the accounts of the intrepid explorers who have reached the area known as Copper Mountain, near the mouth of Coppermine river and Bathurst inlet, a few miles from the Arctic coast, agree on these facts: the geological formation there is similar to that of the famous Keweenaw peninsula already alluded to; this region has for centuries been the source of supply of native copper to the Eskimos and Indians in northern Canada, just as the Keweenaw peninsula in prehistoric time supplied the Mound-builders, and later the Indians; the winter temperatures are but little, if any, more extreme than those experienced throughout much of northern Ontario, Manitoba and Saskatchewan; and this particular area, for 300 miles westward from Bathurst inlet, is of great promise for future detailed prospecting.

We can, after these glimpses, well afford to stand off for a few moments in order to try to estimate even more broadly the probable significance to mining of this enormous region surrounding Hudson Bay, with an area of 2,000,000 square miles, occupying considerably over one-half the entire surface of Canada. Profound erosion, resulting from the attack of geological agencies acting through eons of time, has eaten into and carried off all but the roots and stumps of the vast mountain system which was the first to be pushed above the surface of the primeval ocean. Recent glaciation on an equally vast scale has further worn down and polished the surface, scoured into the softer formations, rounded off into monotonous hills the harder structures, carried away most of the detritus, and exposed the rock formations over an area equalled nowhere else in the world. As I have roughly outlined, a little is known of the general geology and ore deposits of scattered areas, totalling not more than ten per cent of it, and in this smaller area the detailed geology has been worked out over only a few hundred square miles, near important discoveries. But beyond this small fraction only a few intrepid explorers have crossed the Archean area here and there, following the lakes and streams. These agree that the general age and formation of the rocks revealed along these routes of exploration are similar to those already examined in

greater detail. But beyond these broad statements not even general outlines of the geology of most of this vast territory have been accurately worked out. Fortunately, enough is known to make reasonably certain that the broad average formations of the less known parts will closely resemble those of the part already more fully examined. In the only parts where any detailed geological examination and prospecting have been carried out, there have already been discovered the fabulously rich deposits of iron, copper, nickel, gold, and silver already alluded to, containing many thousands of millions (billions) of dollars in gross value of these metals.

A few years ago we heard more discussion about estimating proven ore reserves than about applying knowledge and reason to finding ore. No doubt these discussions did some good, but they had also a somewhat deadening effect. Now, the mining engineer who does not look beyond the ore he can measure, who is not intent on using ascertained facts as a stepping stone to probabilities, will not get very far. How far we can safely step in a case such as that under discussion is a matter of judgment, depending partly on the extent and relative areas of the known and the unknown; partly on the general similarity of the essential conditions of the unknown to the known; and partly on the amount of reliable detailed information which the local experience of the one who is venturing the prediction has given him.

The partly geologized and prospected area of the great Laurentian Shield as outlined is on so large a scale relatively to the whole, and the area of the whole is so vast; also, the similarity of formation, as far as observed, of the wholly unprospected area to the partly prospected area is so marked; as reasonably to preclude the element of chance and to leave the conclusion almost irresistible, that, if geological work and prospecting in detail are persistently and intelligently followed up, results broadly similar to those referred to will repeatedly be obtained in the vast, almost entirely unexplored, and wholly unprospected, areas of this great pre-Cambrian region. And not only this; but continual discoveries in the partly prospected region are convincing evidence that even in this restricted area a mere fraction of the valuable metalliferous deposits has so far been discovered. Only rough prospecting

has so far been carried out over a part of this smaller area. Scientific prospecting is not really begun. In this area, very great possibilities undoubtedly await the future application of science to detailed prospecting, such as is now well begun in Sweden and Finland. In fact, I know of no field of greater promise for the applied scientist than that of prospecting.

I have lived eighteen years in this great mineral region, during which time I have been intimately concerned with the development of one of the first half-score of the great, non-ferrous, metalliferous areas of the world. During this time I have had occasion to keep in touch with the geologizing, exploration, prospecting, and development carried on in that great Northland. In stating these gradually formed convictions, I am expressing what was at first merely a hope; what grew, with study, observation, and experience, to become a probability; and what has finally grown with me to be a practical certainty. When Nature transported the soil from this half of Canada to situations climatically more suitable for agriculture, she balanced the loss to future populations by making possible the discovery of inconceivable mineral wealth, which will amply compensate for the greater faith, larger risks, more far-seeing enterprise, and sturdier hardihood, necessary for its discovery and development. Nature generally does not easily reveal her secrets, but in this instance she has left the door half open.

Now, having dealt in this broad, sketchy way with the mining possibilities of Canada's greatest physical feature, the Laurentian Highland, I hope not to weary you by still briefer reference, from the same standpoint, to her Pacific Highland. This huge area, stretching from the 49th parallel to the Arctic ocean, a distance of about 1,500 miles, and extending from the Pacific ocean and Alaska to the Rocky Mountain foot-hills, a distance of about 400 miles in the widest part, has an area of 550,000 square miles, or a little greater than one-fourth of the Laurentian Highland. It comprises almost the whole of British Columbia and the entire Yukon Territory, the territory having a little more than half the area of the province. In part of this great, mountainous area, lumbering will last for a time and agriculture will be carried on in restricted valleys. But no one in his right senses will challenge the statement

that mining is here also pre-eminently the great basic industry of the future, as it is of the present. This vast territory already has to its credit numerous gold placers, including the famous Klondike; many copper mines, or groups of mines, in which the precious metals are, or were, a by-product — Anyox, Britannia, Sunloch, the Mother Lode, Phoenix, Rossland, and others; a number of lead-zinc mines or groups with silver as a by-product — Sullivan, Slocan, the Lardeau, St. Eugene, and others; gold-quartz mines, of which the best known are Hedley and Ymir; also deposits of iron ore and of a number of other minerals or metals, as for example magnesite and platinum. There are also very numerous prospects, a considerable number of which are just reaching the producing stage. I omit mention of coal resources at this point.

Now here again, in order to clear our minds and keep from being buried in details, we need to stand off far enough to see great facts in proper proportions. The various geological upheavals that have formed the Pacific Highland of North America, and the subsequent erosion, exposing, over this enormous area, formations that would otherwise have remained hidden under their own decay which we ordinarily know as soil, have pre-determined the greatest industry of this area to be mining. Approximately one-third of the total length of this enormous continental back bone and one-fourth of its total area, which equals roughly that of the great Laurentian Highland, is in Canada. Does anyone know of any sufficiently sound reason, even after allowing for certain known differences in glaciation and geological formation, for thinking that Canada's large share of this great area will prove to be less rich in minerals than that falling to Mexico or to the United States? I have diligently searched such evidence as is available, and have tried to be open-minded on the question. I can only say that, again, the areas compared are on such a large scale as practically to eliminate special and narrow differences, and to make the broad conclusion reasonably certain, that Canada's share in this great storehouse of minerals will not prove to be less than her areal proportion of the whole. Mexico has produced metallic wealth for centuries. The United States began production from this

area about the year 1849. Canada entered this western field last, but the widely scattered producing areas, some of which I have mentioned, point to the general conclusion as stated. It is true that contrary views have been expressed, but on evidence that seems to me to be quite insufficient. Similarly pessimistic opinions regarding northern Ontario have been passed from time to time. The essential need in both cases is really scientific prospecting, carried out on a large scale. The easy finds, the great outcrops distinguished by conspicuous gossan, are usually found in the early stages, when the new territory is being rapidly run over. Then comes a pause. Later, real prospecting begins, when a new type of prospector is needed.

These two great regions of Canada, the Laurentian and the Pacific Highlands, occupying, as they do, considerably over two-thirds of the total surface of the entire country, would be sufficient in themselves to place mining in the very forefront of Canadian industries in future. But there are other facts having a most important bearing on the question, and these I shall enumerate as briefly as possible, consistent with leaving a clear impression.

The Appalachian Highland of the eastern United States is continued into Nova Scotia, New Brunswick, and eastern Quebec. For convenience we refer to the part in Canada as the Acadian Highland. I have already referred to the minerals and metals produced in the Eastern Townships of Quebec and the Gaspé Peninsula, which are part of the Acadian Highland. Glaciation has laid bare a large part of Nova Scotia and folding has exposed other parts, so that metal production will always have a large place in the industries of this province, particularly if the Wabana iron ore deposit, owned by the British Empire Steel Corporation, whose blast furnaces and steel works are at Sydney, is included. The enormous resources in ore and fuel of this Corporation, together with its strategic location, must give it one of the very first places in iron and steel production, and even omitting the other large mineral resources of Nova Scotia, give this province a high place in the industries of mining and metallurgy.

But I have so far purposely omitted Canada's most remarkable mineral resource of all, a resource in which she

stands second only to the United States — her practically illimitable coal reserves. Many splendid papers regarding these have been read before this Institute, but I feel that, considering the purpose of the present discussion, I cannot omit at least a summary of the most striking facts. Canada's coal reserves are so vast that I have selected an almost inconceivably large unit of measurement — one thousand million (one billion) tons — roughly equal to thirty years' consumption at the present rate for the entire country. All classes of coal, including lignite, are grouped together. In each case the nearest whole billion of tons is taken. Fuller details are in Dr. Dowling's excellent summary of "The Coal Resources of the World", given to the Institute in 1914, or in the remarkable compilation by the Twelfth International Geological Congress, which visited Canada the previous year, to which Dr. Dowling contributed the article on Canada's coal reserves.

From the latter striking report, which is the best estimate yet made, I cull these startling figures for Canada's estimated total coal reserves:

Unit, one billion tons.

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| Maritime Provinces (Chiefly Nova Scotia) | 10 |
| Prairie Provinces (Chiefly Alberta) | 1,132 |
| British Columbia | 76 |
| Yukon Territory | 5 |
| North West Territory (Mackenzie river basin, north of Alberta) | 5 |
| Arctic Islands | 6 |
| Total | 1,234 |

This inconceivably large figure, 1,234 billions of tons, roughly 35,000 times Canada's present total annual consumption, baffles the imagination. It is nearly sixty per cent greater than the total estimated reserves of the whole of Europe; nearly equal to those estimated for the whole of Asia; and one-sixth of those of the entire world. Can we form any conception of the future value of these reserves, as our country becomes reasonably populated and our various indus-

tries develop? Could any one have conceived the future value of the coal of Europe a century ago? A very small fraction of Canada's coal reserves, and less iron ore than at Wabana, were of sufficient value to excite the cupidity of the industrial magnates of the most military nation in Europe and to become the economic cause of the World War. I mention this merely to show how such assets come to be regarded as population and industries grow.

The coal formations of the entire country are estimated to cover 100,000 square miles, of which about 90,000 square miles occupy territory considered as agricultural. Time forbids making more than passing reference to oil, oil shales, tar sands, gas, gypsum, salt, building stone, and other minerals and structural materials, of great value, widely distributed over much of our agricultural area. There are strong reasons for believing that oil, probably in great quantities, exists in the Great Central Plain, in the Mackenzie river basin, beyond the climatic limit of profitable agriculture. Very large areas of tar sands also are known within this area.

Let us again pause and, by making a brief summary of these facts, try to grasp, to realize, grip and weigh the relative value of these fundamental facts regarding Canada's mineral resources in something like their true proportions. This is particularly necessary, because most Canadians live in our agricultural areas, and because of this have either little knowledge, or no keen realization, of the greatest of our natural resources.

In the first place, the total area of Canada, almost 3¾ million square miles, is approximately equal to the total area of the United States and Alaska, including all dependencies, and is only 6 or 7% less than the area of Europe. Of this vast area nearly 70% is occupied by the great Laurentian and Pacific Highlands, which, because of recent glaciation, mountainous surface, or unsuitable climate, are fit for profitable cultivation only in restricted valleys, totalling probably considerably less than 3% of their entire surface. Between these two Highlands lies the Great Central Plain, enormously rich in coal, probably rich in oil also, the southern part of which, lying in the Prairie provinces, comprises 70% of the entire profitably cultivable area of Canada. But note particularly

that even this Great Plain, to which may be added several of the Arctic Islands, is known to be enormously rich in mineral resources, particularly in fuel.

This leaves only southern Ontario, comprising less than 10% of the province; southern Quebec, embracing less than 10% of this province also, a considerable part of which is occupied by the Acadian Highland; and the Maritime Provinces, rather less than two-thirds of which, 30,000 to 35,000 square miles, is suitable for profitable agriculture.

Adding together all these areas, suitable in surface, soil, and climate for profitable cultivation, including the cultivable valleys in the Highlands, we find a total of rather less than 15% of the area of Canada suitable for profitable agriculture. This leaves 85%, of which the principal or only permanent basic industry is or will be mining. But fully one-third of even this agricultural part of Canada is known to be enormously rich in minerals.

Over a large part of this mining area, lumbering will for many years be an important industry, and over some of it re-forestation may make lumbering a permanent industry. But exclusive of this factor, which, so far, has made all too little progress, we shall find that as the lumberman succeeds the trapper, so, speaking broadly, will the prospector and the miner succeed the woodsman.

The miner is not limited by climate as is the agriculturist. Wherever the climate of any region is not so severe as to leave it under a perpetual mantle of snow and ice; and wherever, because of glaciation, geological formation, or ordinary erosion, there is reasonable rock exposure, there will men persist in seeking mineral treasures. Even where these rock exposures do not frequently occur, but where stratification and folding can be worked out so as to lead to reasonable suspicion of hidden mineral, such as gas, oil, salt, sulphur, or coal, there will men of enterprise risk their money in exploring with the drill.

In this discussion, I have purposely avoided statistics and technical expressions. It is most important that these facts, which are so vitally related to the future development of Canada, should grip the mind of the Canadian public with full significance. No text book that I have examined, and

no scientific article that I have read, brings out with anything like due proportion and emphasis the enormous importance of the very simple facts and inferences to which I have called attention. Even the *Encyclopaedia Britannica* begins its article on Agriculture in Canada with the statement, "Canada is pre-eminently an agricultural country"; and later in the article dilates on an agricultural belt as wide as France stretching across the country for 3,500 miles. The author of this article can hardly have been an observant traveller. If he had said, "Canada's present chief basic industry is agriculture", no exception could be taken to the statement. But to write of a country, only about 15% of which is suitable for profitable agriculture, 85% of which must and will find its greatest economic future in its minerals, and 90% of which will undoubtedly produce vast mineral wealth, as being pre-eminently agricultural, is to show a lack of a proper sense of proportion, because of enthusiasm for the subject in hand.

Our country cannot make the most rapid and satisfactory advance by over-emphasis of part of our natural resources. There are sound reasons, as I have pointed out, for delay and caution in making any forecast of mineral resources. But we have now reached a stage at which, in my judgment, reasonable forecasts are not only possible and desirable, but necessary, for Canada's best economic advancement.

Canada has probably less than one-third of the area of land suitable for farming, available in the United States. In that country there are nearly six and one-half million farms, with an area totalling nearly one billion acres, supporting a farming population of a little over 30 millions. No doubt, irrigation, more clearing, and more intensive cultivation, will increase this population, possibly to 60 millions, before discomfort will result. On a similar basis our agricultural lands, when fully settled, should support ten or twelve millions, and under continued improvement and more intensive cultivation may support a farming population of 20 or even 25 millions in comfort and with profit. These figures are for population engaged in agriculture only. If our total population bore to the rural the ratio presently obtaining in the United States, we would have a total of about 3⅓ times these figures.

Now these are facts and estimates of very great importance indeed. But there is no reason for allowing them to obscure the immensely important facts and inferences concerning Canada's potential wealth in minerals; nor should it cause us to lose sight of the relative importance of basic industries, so that we may carelessly alienate to foreign owners too many of our priceless assets, before we ourselves properly realize their value.

It must be apparent that too little attention has been paid to our mineral resources. This is perhaps because of the natural succession of basic industries to which I have already alluded: hunting and fishing, lumbering, farming, mining — which, with some over-lapping, is the usual order of development of these industries. This Institute can perform a very valuable public service by systematically bringing to the attention of the public the great importance of Canada's mineral resources, and by directing the attention of young people to mining as a future life's work.

In conclusion, I wish to speak in the highest terms of the work already done by our Dominion and Provincial Governments, and of the very great assistance to mining which the devoted members of the various Geological and Mining Departments have rendered. But I also wish to invite the attention of these Governments to the future relative importance of mining and agriculture; to request that they reconsider this question in view of our steadily increasing knowledge; and to ask, respectfully, whether mining is receiving its relatively fair share of government assistance.

For discussion on Dr. Corless' address, see *Monthly Bulletin* (1922) for May (p. 612), June (p. 709), August (p. 921), September (p. 1003), October (p. 1093) and December (p. 1269).

# ONTARIO GOLD DEPOSITS[1]

## Their Character, Distribution, and Productiveness.

By Percy E. Hopkins.

(Annual General Meeting, Ottawa, March, 1922.)

### Introduction.

Gold-mining, at the present time, is receiving renewed attention. The abundance of labour, the reduction in mining costs, and the premium on the metal, are attractive features in this industry. In this paper will be given a historical summary of gold mining in Ontario, together with notes on the production, general character of the deposits, and other features.

### History.

Since gold was first discovered in Ontario in 1866, at the Richardson location, Hastings county, southeastern Ontario, efforts have been made from time to time to establish the industry. Permanent success was not met with until Porcupine was discovered in 1909. In southeastern Ontario gold has been produced spasmodically up to the present from numerous properties, the principal ones being the Deloro and Cordova. The first important discovery in western Ontario was made by Peter McKellar in 1871 in what is now known as the Huronian mine, Moss township. In 1872 gold was found on an island in Partridge lake. This was followed by many other discoveries in the Lake of the Woods district, where the chief mines were the Sultana, Mikado and Regina. In 1875 small

[1] This paper has since been published in fuller form as Part 2 of the Annual Report (Vol. XXX) of the Ontario Department of Mines. Accompanying the Report are various maps showing the location of the producing gold mines and many prospects. Railway stations and wagon roads serving the gold properties, together with hydro-electric power plants and transmission lines related to the mining industry are shown. Mining claim numbers are also given in the text, together with many illustrations and numerous references to the literature.

nuggets of gold were found on Victoria cape, near Jackfish, Lake Superior. The discovery of gold at Rainy lake in 1892 resulted in prospecting soon being extended into Shoal lake, Seine river, Manitou, Sturgeon lake, and other neighbouring areas. The gold boom of western Ontario reached its peak in 1897, resulting in many unwarranted stamp mills being erected. Inefficient management and stock-jobbing operators caused the loss of much capital; nevertheless certain mines — the Sultana, Mikado and Regina — produced about one-half million dollars each.

Gold associated with copper was found at the McGown property, two miles east of Parry Sound, in 1894, but little production resulted. In 1895 and 1896 the precious metal was discovered at Jackfish and Schreiber, respectively, and the area was enlarged in 1914 by the discovery of gold at Duck lake, 15 miles north of Schreiber. From 1892 to the present time, isolated properties situated near Wanapitei and Sudbury, and westward along the north shore of Lake Huron, have had a small production; these include the Crystal, Havilah or Ophir, Shakespeare, Long lake, and others. In addition, a small quantity of gold is recovered from the nickel-copper ores at Sudbury. Since 1902, three or four mines have produced in the vicinity of Wawa, Michipicoten, where gold was found in 1897, but they are now lying idle. Of these, the Grace is the best known. East of Goudreau, on the Algoma Central railway, gold was found in 1918; in 1921, spectacular showings were made on the Murphy claims, southwest of that station.

The discovery of Cobalt in 1903 brought hundreds of prospectors into northeastern Ontario. From that year up to 1918, important gold areas were found annually. They are as follows: Playfair, 1905; Abitibi and Larder Lake, 1906; Night Hawk Lake, 1907; Painkiller, 1908; Munro and Porcupine, 1909; Swastika, 1910; West Shiningtree, 1911; Kirkland Lake, 1912; Boston Creek, 1914; Kowkash-Tashota, 1915; Matachewan, 1916; Rickard and Lightning River, 1917; Bourkes, 1918. In addition to these, many small gold discoveries have been made in other townships. Of these areas, Porcupine is by far the largest producer, the principal mine, the Hollinger, being one of the great gold mines of the world.

The Kirkland Lake area is next in importance. Ontario now leads the other provinces of Canada, and every state in the United States, California excepted, in gold production, the present yield being 75% of the total for the whole Dominion. Gold is so widespread, and the output is so substantial and increasing so rapidly, that it seems fitting to name Ontario the *'Golden Province'*.

## PRODUCTION.

The total gold production of Ontario up to the end of 1920 was $73,153,271, all of which came from pre-Cambrian deposits. From 1892 to 1911 the production was small, the greatest output for any year being $424,568 in 1899. During these years the Sultana, Mikado, and Regina were the principal producers. Gold has been recovered as a by-product from the copper-nickel ores mined at Sudbury, where work began in the late eighties. A small recovery of gold has also been made from the Cobalt ores. The Lake of the Woods mines were not so active between 1904-1911. In 1909, the Laurentian was the only producer.

TABLE 1. GOLD PRODUCTION OF ONTARIO.

| Year | Ounces | Value | Year | Ounces | Value |
|---|---|---|---|---|---|
| 1866–1891 | 9,200 | $190,258* | 1907 | 3,810 | $66,399 |
| 1892 | 1,785 | 36,900 | 1908 | 3,465 | 60,337 |
| 1893 | 1,695 | 32,960 | 1909 | 2,042 | 32,445 |
| 1894 | 2,022 | 32,776 | 1910 | 3,619 | 68,498 |
| 1895 | 3,030 | 50,281 | 1911 | 2,062 | 42,637 |
| 1896 | 7,154 | 121,848 | 1912 | 102,278 | 2,114,086 |
| 1897 | 11,412 | 190,244 | 1913 | 220,837 | 4,558,518 |
| 1898 | 16,261 | 275,078 | 1914 | 268,942 | 5,529,767 |
| 1899 | 27,594 | 424,568 | 1915 | 411,588 | 8,501,391 |
| 1900 | 18,767 | 297,861 | 1916 | 497,833 | 10,339,259 |
| 1901 | 14,293 | 244,443 | 1917 | 420,893 | 8,698,735 |
| 1902 | 13,625 | 229,828 | 1918 | 411,878 | 8,502,480 |
| 1903 | 10,383 | 188,036 | 1919 | 505,964 | 10,451,709 |
| 1904 | 2,285 | 40,000 | 1920 | 564,312 | 11,665,851 |
| 1905 | 5,541 | 99,885 | | | |
| 1906 | 3.926 | 66,193 | Total | 3,568,496 | $73,153,271 |

*Approximate.

Up to 1911, gold-mining in Ontario was not profitable on the whole. Since then the industry has flourished, owing

particularly to Porcupine, and, to a lesser extent, to Kirkland Lake. The 1913 production, $4,558,518, almost equals the entire output up to that year from the time gold was first found in the Province in 1866. In 1920, the gold production exceeded the silver output for the first time since Cobalt was discovered in 1903. It is estimated that the gold production of Ontario for 1921[1] will be worth about fifteen million dollars. In every important gold producing country, with the exception of Canada, the output has steadily declined in the last four years, and the gold mines of Ontario stand out as the only ones which materially increased their output throughout that period. Ontario has promise of being a much larger producer. The world's maximum production of 468.7 million dollars in 1915 has steadily decreased to 339.4 million dollars in 1920. Table 1 shows the yearly and total gold production of the Province up to the end of 1920, as published by the Ontario Department of Mines.

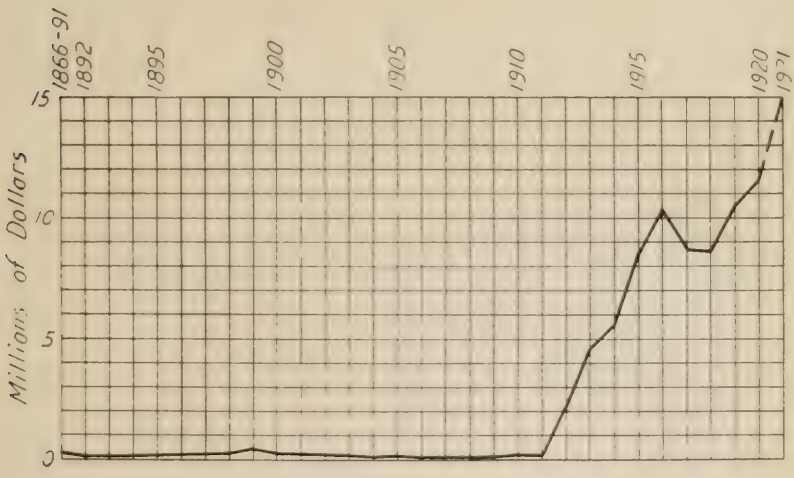



DIAGRAM SHOWING GOLD PRODUCTION OF ONTARIO

[1] During the first nine months of 1921 the gold production was valued at $9,818,073.

**LEGEND**

1. Lake of the Woods and West Shoal lake.
2. Eagle lake, Wabigoon or Dryden, Manitou and Sakoose.
3. Sturgeon lake.
4. Mine centre, East Shoal lake or Lower Seine area.
5. Atikokan or Upper Seine area.
6. Huronian and Shebandowan lake.
7. Kowkash and Tashota.
8. Jellicoe, Kinghorn and Little Long lake (only a few assays have been obtained from this area).
9. Schreiber, Duck lake and Jackfish.
10. Michipicoten and Goudreau.
11. Ophir or Havilah. Galbraith township).
12. Shakespeare, Howry creek and Whiskey lake.
13. Long lake and Sudbury.
14. Tionaga.
15. Porcupine, Cripple creek, Night Hawk lake and McArthur township.
16. Patten river (only assays obtained here).
17. Munro, Painkiller lake, Rickard, Abitibi lake and Lightning river.
18. Kirkland Lake, Larder Lake, Boston Creek, Swastika and Bourkes.
19. Matachewan.
20. West Shiningtree.
21. Timagami.
22. Wanapitei.
23. Parry Sound.
24. Southeastern Ontario (Peterborough to Lanark county).

**Lode Gold Areas of Ontario.**

## PRE-CAMBRIAN GEOLOGY OF ONTARIO.

The known gold areas and the vast extent of the pre-Cambrian rocks of Ontario are shown on the accompanying geological sketch map. Much of the surface underlain by these rocks is unprospected and worthy of more attention from those in search of gold deposits. The pre-Cambrian structure is so complex that many years must elapse before detailed maps of the whole can be prepared; however, the most promising gold-bearing rocks of the Province are being gradually mapped. The age classification used in this paper is that employed by the Ontario Department of Mines, and is as follows:—

*Pre-Cambrian*

- Keweenawan
- Animikie
- Algoman
- Haileyburian
- Timiskamian
- Laurentian
- Loganian
  - Grenville
  - Keewatin

Gold occurs in rocks of the various systems, particularly in basalt schists belonging to the Keewatin, and in Timiskamian sediments that have been intruded by Algoman granite and porphyry. Examples: Hollinger, McIntyre, Dome, Tough-Oakes, etc. Most of the gold at Kirkland Lake, however, comes from Algoman porphyry, although, in places, the wall rock is a syenite. A small production has come from veins in the granite; examples: Foley, Elizabeth and Harold Lake. The main vein on the Regina, Mikado, and St. Anthony, passes from altered granite into Keewatin schists.

## GOLD EPOCHS.

There were probably four epochs of gold deposition during pre-Cambrian times, namely (a) Keweenawan, (b) Algoman,

(c) Haileyburian, and (d) Laurentian and Keewatin [1]. It is often difficult to determine the epoch in which an ore body was formed. Hence, in the following pages, the age given for certain deposits is to be considered only provisional.

*Keweenawan.*—The gold deposits connected with the Keweenawan intrusives are relatively of little importance. The following are provisionally classed as being of this age: the Crystal mine on Wanapitei lake; the Havilah, formerly the Ophir, mine, north of Thessalon; the Sudbury copper-nickel deposits, of which the Vermilion mine has the richest gold showing; the Bruce copper mines, which carried small quantities of gold in places, as do also certain cobalt-silver ores; portions of the deposits at Howry creek and Whiskey lake, and the Ogema and other Keweenawan lead-zinc deposits in the vicinity of Black bay, which carry small quantities of gold and silver.

*Algoman.*—All the known important gold deposits of Ontario appear to have been formed during this epoch. The deposits at Porcupine, Kirkland Lake, Boston Creek, Matachewan, Deloro township, and elsewhere, are believed to belong to this epoch.

*Haileyburian.*—Certain gold deposits appear to be genetically related to intrusives of this age, namely, the Long Lake gold mine, the Cordova, the Jameson in Robb township, and the Shaft island deposit on Abitibi lake.

*Laurentian and Keewatin.*—Following the intrusion of Laurentian rocks, doubtless related gold deposits were formed, but they have been removed, for the greater part at least, by erosion. It is difficult to prove that deposits referred by certain writers to this epoch are not of later age. The term Laurentian has been often loosely used; the same may also be said of Keewatin. The Foley and other mines at Mine Centre are genetically connected with an altered granite or protogine described as Laurentian by A. C. Lawson, although A. L. Parsons refers to the rock as Algoman. The Ophir, Sultana, and Sawbill occur in a porphyritic gneiss which may be Laurentian. The Goodfish lake deposits are related to schistose quartz-porphyries which may be of Laurentian age. The fact

[1] Ont. Bur. Mines Report, Vol. XXIV, pp. 243-248.

that gold occurs in small quantities in the Keewatin rhyolites at Lightning river and other places would suggest that some of the gold may be of Keewatin age.

## TYPES OF GOLD LODE DEPOSITS.

The productive gold deposits of Ontario with which this paper deals are primary or lode deposits. Placer gold occurs on the Vermilion, Wanapitei and Grassy rivers, Meteor lake, Lake Manitou and Lake Savant, and in various townships such as Munro, McCool, Montrose and Timmins. Attempts at placer mining so far have proved unsuccessful, but little systematic work has been done.

1. *According to minerals present* the lode deposits may be divided into five classes, namely, (a) the *pyrite-gold-quartz type* in which pyrite is the predominating sulphide. This is the most common and most productive type, as represented by the Porcupine deposits. (b) The *arsenopyrite-gold-quartz type* in which arsenopyrite is prominent. The southeastern Ontario, Timagami, Long Lake, Howry creek, and some of the Painkiller lake deposits are the only known ones belonging to this type in Ontario. (c) The *gold-telluride veins*, characterized by the presence of tellurides. Calaverite and petzite, gold tellurides, are found at the Miller Independence; both calaverite and kalgoorlite occur in the No. 3 vein of the Tough-Oakes; calaverite occurs on the Boston-McRae, and petzite has been recognized at the Bourkes mine. Gold-bearing tellurides probably occur at various mines in Kirkland lake and other places, but they are so finely disseminated that they are difficult to recognize. However, altaite is quite common, while coloradoite, tetradymite, and hessite occur in small quantities. Several other tellurides have been found in different parts of the Province, and although they are non-gold tellurides they frequently indicate rich gold ore, since native gold invariably occurs as veinlets in the tellurides. (d) *Calcite-gold type.* Calcite is the principal gangue mineral in the McQuaig and Sjolander-McKirdy deposits at Big Duck lake. Some of the veins on the Argonaut (Huronia) are made up largely of calcite. (e) Some gold is recovered as a by-product from the copper-nickel ores of Sudbury.

2. The shape of the deposit is of small importance since most of them are irregular in form; however, *according to form or structure* the deposits may be classified into several groups, following the descriptions of the various authors who have examined them. These groups are: (a) *"true fissure" veins.* Examples: Mikado, Regina, Pine Portage, Foley, No. 1 vein on Miller-Independence, Clifton-Porcupine, and a portion of the Rea vein. (b) *Composite veins or lodes.* Most of the productive deposits are of this type. The Hollinger, McIntyre and Dome deposits at Porcupine are the largest of this type known in Ontario. (c) *Sheeted zones.* Examples are the Howey-Cochenour at Lightning river and portions of the deposits in the Kirkland lake area. (d) *Stockwork.* The stockwork type in granite, porphyry, and dolomite or ankerite is represented by the Ferguson at Mine Centre, Gold island at Night Hawk lake, and deposits at Larder lake. (e) *Fahlbands or shear zones.* Examples are the Scramble, Dome lake, the Currie at Boston Creek, and the Elstone-Dunkin in Gauthier township. (f) *Pyrite and iron formation,* comprising alternate layers of chert, sugary quartz, magnetite and pyrite, which frequently carry a little gold. Where secondary quartz veins intersect the iron formation, gold is more prevalent. The Big Master vein shows a remarkable similarity to iron formation. The quartz of the Elizabeth at Atikokan is white and granular, like silica formed in iron formation. Other examples are the Wright, James, Philadelphia, and Red Dog deposits near Porcupine. (g) *Single lenses and pipe-like deposits.* Portions of the Cordova deposits are pipe-like. The Regina and Mikado have a nearly vertical chimney or short shoot. The Long Lake deposit has the shape of a large vertical core or pipe. Many ore deposits are single lenses, like the Croesus.

## BRIEF DESCRIPTION OF GOLD MINING AREAS.

### *Porcupine* [1].

This is by far the most important gold area in Ontario. The first reported gold discovery was made in 1908 in quartz

[1] Porcupine Gold Area, Ont. Bur. Mines Reports, Vol. XX, Part 2, pp. 1-39; Vol. XXI, Part 1, pp. 205-249; Vol. XXIV, Part 3, pp. 1-57, by A. G. Burrows.

and schist on the Hunter claim on Porcupine lake. The chief find, which caused a rush to the area, was made by J. S. Wilson in 1909 on the Dome mines. During the following two years several mills were under construction and production began. Up to the end of 1920 the total production of gold is valued at $64,737,349, and the ore reserves approximately $50,000,000. During 1920 the production was $10,597,572; in 1921 it is likely to be in the neighbourhood of $13,700,000. The three principal mines are the Hollinger and McIntyre, near Timmins station, and the Dome, three miles to the southeast. The ore deposits occur in Keewatin basic volcanic schists in the Hollinger area, and in the Keewatin volcanics and Timiskamian sediments at the Dome; in both cases they are near the contact of schistose quartz-porphyry, which is classified as pre-Algoman in age. According to A. G. Burrows, the Ontario Department of Mines' geologist who studied the ore deposits, the intrusions of quartz-porphyry at different parts of the area have in some way influenced the deposition or location of the gold, but it is not likely that the porphyry has been the source of the gold-bearing solutions. W. G. Miller regards the quartz veins as genetically related to the Algoman granites and related rocks of the region.

The ore deposits are quite large in size, low to medium in grade, and of the lode type in structure, containing much mineralized schist. Iron pyrites is the most abundant sulphide; chalcopyrite, galena, and zinc blende usually occur in the richer parts. Pyrrhotite is quite common at the Dome. Other minerals associated with the schist and quartz gangue are calcite, dolomite, scheelite, tourmaline, graphite, feldspar, chlorite, sericite, etc. No. 5 vein on the McIntyre has an ore shoot 1,500 feet in length; the same ore shoot continues for at least 100 feet on Hollinger ground. No. 1 vein on the Hollinger will average 10 feet of width in ore for a length of 1,000 feet; while No. 84 vein has an ore shoot 900 feet long with a lean section in the centre. Many shoots are 500 feet in length. The value of the ore has not decreased in depth. Much faulting has been encountered, more particularly in the upper levels of the McIntyre No. 5 vein and on the Porcupine Crown vein.

Following are brief descriptions of the ore deposits and of the various types of gold occurrences; these are largely summarized from Mr. Burrows' reports. Reference should be made to the maps accompanying Mr. Burrows' report for the geology and locations of these various deposits. Many gold finds of greater or less importance have been made in almost every township in the vicinity, but these are too numerous to enumerate; however, the several types will be mentioned. Production figures for mines mentioned hereunder are complete up to the end of 1920.

*Hollinger.*—The Hollinger, which is by far the greatest gold mine in Canada, comprises 400 acres near the town of Timmins in Tisdale township. All the rocks are schistose, consisting of Keewatin ellipsoidal basalt and dacite intruded by wedge-shaped chonolithic masses of quartz porphyry. The apex of the main, canoe-shaped mass of quartz porphyry around which the ore bodies occur recedes or dips approximately 50° to the northeast. In the green schist, and surrounding this quartz porphyry mass, are numerous, large, closely-spaced, nearly parallel veins or lodes comprising quartz and mineralized green schist. Some of the veins extend into the porphyry, but the gold values in the veins in the porphyry are either low or erratic. The veins cut the schist at a low angle, and appear as overlapping lenses, many of which are connected by branch veins presenting large lode deposits. The southerly veins dip 87° to the southeast, while the big No. 84 vein on the north side of the porphyry dips 85° to the northwest. Underground workings total 40 to 45 miles, the deepest level being 1,530 feet. In 1920 sinking, drifting, cross-cutting, and raising amounted to 18,103 feet. The electric haulage system is used on the main levels underground. During the latter part of 1921 the mill treated approximately 4,000 tons of $8 ore daily. The total production and dividends paid to the end of 1920 are $39,142,649 and $13,360,000 respectively. In the annual report of the Hollinger for the year ending Dec. 31st, 1920, the developed ore is valued at $37,693,450. During 1922 it is expected that the mine will be a large low grade property with a nearly uniform and maximum production.

*McIntyre.*—The main quartz porphyry mass and veins Nos. 64, 84 and 91 of the Hollinger extend northeasterly to the

McIntyre (which has taken over the Pearl Lake and Jupiter mines); these vein extensions on the latter property are known as Nos. 1, 7 and 5 respectively. The northerly No. 5 vein, which dips highly to the northwest, has an extremely long ore shoot (1,500 feet) which rakes steeply to the northeast. Much graphite is present in parts of the deposit. Vein No. 7, which lies a short distance south of No. 3, has been opened up for a considerable length on some of the lower levels. Vein No. 1 on the south side of the porphyry was also an important deposit. Owing to the porphyry mass being wedge-shaped, more green schist and additional ore bodies are being encountered on the lower levels. In all, about 63,000 feet of work has been done. The mill treats 550 tons of approximately $11 ore daily, and is being enlarged to 800 tons capacity. The total production to the end of 1920 was $10,041,785 in gold and $92,033 in silver. Dividends paid up to the end of 1920 amounted to $1,994,656. Developed ore reserves are valued at approximately 5½ million dollars.

*Dome.*—The Dome Mines, with which is incorporated the Dome Extension, comprises 400 acres lying 3 miles southeast of the Hollinger and McIntyre. The ore bodies are very similar to those of the Hollinger, occurring in Keewatin basalt and Timiskamian sediments, and largely on the northerly edge of a schistose quartz porphyry mass. Some ore comprises well mineralized schist with numerous rich quartz veinlets. The veins strike a little north of east and dip nearly vertically, presenting a large low grade, egg-shaped deposit, pitching about 45° on to the Dome Extension. Big low grade deposits have been worked by means of a large open pit some 250 feet in width and 1,000 feet in length. Recently the individual lodes are being worked from 10 to 40 feet or more in width, yielding a higher grade of ore. Pyrite, pyrrhotite and galena are the chief sulphides. Ore of a good grade has been developed on the deepest level, namely, 1,300 feet. Electric motors with storage batteries are in operation on the main levels. The mill has recently been treating about 1,000 tons of $7 ore daily; and sufficient ore is blocked out to keep the mill running at a maximum capacity for 3 or 4 years. The total production, 1912 to 1920 inclusive, has been $11,541,376

in gold and $63,892 in silver. Dividends paid to the end of 1920 were $1,916,886.

*Porcupine Crown.*—This mine to end of 1920 produced $2,863,873 from a 5 to 7-foot quartz vein which strikes north and south and dips steeply to the east on or near a porphyry-basalt contact. The property, together with the two Thompson-Krist claims, is now known as the *North Crown Mines.*

*Vipond-North Thompson.*—The veins in this mine occur in altered greenstone. They have been worked to a depth of 600 feet, the production being $808,204. This has come largely from the Vipond.

*Schumacher.*—On this property there has been a small production ($561,885) from narrow, contorted, low grade veins in highly altered Keewatin schists. The *Rea* also produced a small amount, namely $147,076, from a rich ore shoot, approximately 200 feet deep and 4 feet thick.

Other promising prospects in the Hollinger area are the *Plenaurum, Rochester* and *Moneta.*

*Dome Lake.*—On this property there are small isolated ore shoots, 2 or 3 feet in width, in an east-west shear zone or fahlband, 1,000 feet in length. The vein has been worked to a depth of 500 feet, where the deposit dips north on to the West Dome. The property, which is now closed, has produced to date approximately $321,826.

The *West Dome* and *Apex* also have an ankerite vein some 20 feet in width and cut by auriferous quartz veinlets from which a small production has been recorded.

The *Porcupine Pet, Preston* or *Clifton-Porcupine,* and *Porcupine Porphyry Hill,* in Deloro township, have each produced a little gold from narrow quartz veins in a rather massive quartz porphyry. In the north part of the area there are large east-west shear zones in Keewatin greenstone. These are impregnated with secondary carbonate, quartz and pyrite, and contain large lenses of quartz up to several feet in width. The *Davidson* is the only one of this type in the area which has produced, the amount being $53,914. The *Gold Reef* and

*Tommy Burns* or *Triplex* have each yielded a few hundred dollars in gold from narrow veins in basalt and porphyry respectively. Among other types of deposits in the area may be mentioned the *Keora*, *Anchorite*, and *Maidens-McDonald* in rusty carbonate; the *Three Nations*, *North Dome*, and *McAuley-Bridge* in Timiskamian sediments; the *Wright* in iron formation; and the *Gold Island* in aplite. The *Peninsula* is encountering rich ore in green schist on the 80-ft. level and in the 200-ft. shaft.

### *Kirkland Lake*

Kirkland Lake[1] is the second important gold-producing area in Ontario. Up to the end of December, 1920, the production of gold, with some silver, was valued at $4,021,473. Five plants are now producing $2,000,000 annually. The producing mines are situated along a fault zone 2½ miles in length. In this zone there are irregular low to high-grade ore shoots of various sizes. These are lode or composite vein deposits of the gold-telluride type. The fractured zone contains several faults, some of which are later than the gold, and it crosses the different rocks in its path, including Timiskamian conglomerate, and Algoman syenite, porphyry, and lamprophyre, all the intrusions being differentiation phases from the same magma; the magma is also the source of the gold. The ore minerals are native gold with small quantities of calaverite, kalgoorlite, and hessite. The lead telluride, altaite, is common, while coloradoite and tetradymite have rarely been encountered. Other minerals present are iron pyrites, chalcopyrite, molybdenite, graphite, barite, galena, and zinc blende, in a gangue of the enclosing rock (syenite, porphyry, or conglomerate) with quartz and a little calcite. All the ore is similar in appearance and is reddish in colour, due to the included porphyry. The ore shoots vary in width from 3 to 40 feet and in grade from $5 to $30 or more. The producing mines are briefly described below, commencing at the west and taking them in order to the east along the main zone.

[1] Kirkland Lake Gold Area, Ont. Bur. Mines Report, Vol. XXIII, Part 2, 1914, pp. 1-39; Vol. XXIX, Part 4, 1919, pp. 1-48, A. G. Burrows and P. E. Hopkins.

*Kirkland Lake* mine has a 140-ton mill and is milling ore from a large shoot, which extends from the 250-foot to the 900-foot level, the greatest depth in the area. Other ore shoots are being encountered further west on the lower levels.

*Teck-Hughes* has stoped ore up to 40 feet in width from the 600-foot level nearly to the surface. The ore is put through a 150-ton mill, the extraction being approximately $9 per ton.

*Lake Shore* has two veins 400 feet apart with mine workings 600 feet in depth. The stopes average about 11 feet in width. During 1920, the 60-ton mill treated 19,779 tons of ore yielding $503,735, or $25.47 per ton. The 1921 production was approximately the same.

*Wright-Hargreaves* has deposits quite similar to the Lake Shore and is opened up to a depth of 600 feet. The mill, which commenced operating in May, 1921, treats 175 tons of ore daily, averaging about $12.00 to the ton. The vein on the surface is three-quarters of a mile in length.

The *Tough-Oakes*, with which is amalgamated the *Burnside*, was the first producer in the area. The mine produced $1,933,955 from 3 narrow veins which passed from porphyry into conglomerate, the ore averaging $20 to $75 per ton across 5 feet. Coarse calaverite came from No. 3 vein. The mine was re-opened in 1921, and will probably be producing again in 1922. The capacity of the mill is 125 tons daily. Other prospective producers on the main zone, which have similar looking deposits, are the *Orr*, which is 400 feet deep, and the *Sylvanite*, which is having a shaft sunk to a depth of 300 ft. The *Elliot-Kirkland* has a 500-foot shaft on the westerly end of the main fracture.

About three-quarters of a mile south of the main zone are a number of properties, mentioned in order from west to east: *Canadian Kirkland*, *Honer*, *Hunton-Kirkland*, *Ontario-Kirkland*, *Black*, etc. Of these, the Ontario-Kirkland is the most promising. Ore has been developed to a depth of 450 feet in greywacké schist; a 100-ton mill has been constructed and began producing early in January, 1922. The *Comfort-Kirkland*, which adjoins the Ontario-Kirkland, is crosscutting on the 150-ft. level.

Mineral deposits resembling those of Kirkland Lake are being exploited in Lebel township, 6 miles east of Kirkland Lake. These deposits include the *Bidgood*, where an ore shoot has been encountered in porphyry on the 300-ft. level; the *King-Kirkland*, which has drifted on a short ore shoot on the 100-foot level; and the *Lebel-Oro*, which has a shaft on a narrow rich vein. Promising looking deposits have also been found at Goodfish Lake and Swastika.

For a brief description of the numerous other gold areas in Ontario reference may be made to the report on "Ontario Gold Deposits" which forms Part 2, Volume XXX of the annual reports of the Ontario Department of Mines. Among the present active properties in these areas may be mentioned the Argonaut and Canadian Associated Goldfields at Larder Lake; Miller Independence at Boston Creek; Bourkes at West Shiningtree; Murphy at Goudreau; Jackson and McKellar-Longworth at Schreiber; and St-Anthony at Sturgeon Lake. The Mikado on Shoal lake, Lake of the Woods, is preparing to resume operations.

## DISCUSSION

MR. D. L. H. FORBES: In Mr. Hopkins' paper an excellent outline is given of the history, production, and general character of Ontario's gold mines. It is apparent from recent production figures that, in spite of the fact that it took thirty-five years, dating from the first discovery of gold, for profitable production to be established in the province, the last ten years have witnessed a rapid growth in the Porcupine and Kirkland Lake districts, and capital will be encouraged now to develop gold prospects that would otherwise remain dormant.

With a length of over 700 miles east and west from Quebec to Manitoba, and an average width of about 100 miles, the known belt of gold-bearing pre-Cambrian rocks of Ontario offers an attractive field for mining enterprise, in spite of the numerous failures and abandoned workings that dot the area from end to end.

In regard to the geological age in which the gold deposition took place, the correct determination of which in each district is important in discriminating between favourable and unfavourable conditions for the development of prospects, I think there is at least a reasonable doubt of the correctness of the present belief of the provincial geologists that in all the existing mines gold was deposited only during the pre-Cambrian era. At any rate there is abundant evidence from recent developments at the Teck-Hughes mine in the Kirkland Lake district that ore deposition occurred subsequent to the intrusion of a diabase dike that is supposed to have been formed in the Keweenawan period. One is forced to draw the inference from this evidence that the principal ore deposition occurred after a period of vulcanism that was more recent than the Keweenawan and that produced profound dislocations and intense fracturing of pre-Cambrian rocks over a long period of geological time. If this inference be correct, it would seem probable that there was an extensive upheaval of granitic rock magma adjacent to or under the intrusive rocks and metamorphosed sediments of the pre-Cambrian era in which gold ore is now being mined. A rock of the grano-diorite type will perhaps be found at some future time and prove to be the rock whose upheaval and varying upward pressure while molten was responsible for the enormous faults and numerous vein fractures of the district and whose emanations of mineral solutions and gases when solidifying furnished the vein filling and gold contents of the fractures and crushed zones of rock adjacent to the fractures.

Similar conditions to those at Kirkland Lake, with modifications in accordance with the character of the fractured rocks, probably occurred during the same geological period over a large area of Northern Ontario and it is practically certain that the present list of producing mines will be more than doubled during the next ten years.

MR. H. B. ROBINSON: The Reports of the Ontario Department of Mines constitute the most extensive literature on the gold districts of Ontario. Their geologists have kept pace with, and even anticipated, discoveries. Their reports have maintained a high degree of excellence and have afforded great

practical assistance to prospectors and operators. Owing to the complexities of the pre-Cambrian terraine and to the number of districts opened up in a short period, their work has perhaps leaned strongly towards preliminary reconnaissance, but I look forward to, and I believe the importance of the gold industry justifies, more detailed work. Mr. Hopkins' article is a careful summary of the various reports; the salient points have been well brought out and will form a convenient background for discussion.

In each field there are a number of local problems and it may be interesting to mention a few from the Porcupine district. The quartz porphyry, which is so intimately associated with the ore-bodies, has been classed as pre-Algoman in age, principally, I believe, because it had been rendered schistose previous to the formation of gold-bearing quartz veins which intersect it. It seems to me that the field conditions are not incompatible with the theory that the porphyry is Algoman in age and either the source of the gold-bearing solutions or derived from a common parent magma. The porphyry intrudes all the formations intruded by the more massive granite and feldspar porphyry, which is admittedly Algoman. The schistose condition may be explained on the supposition that it is not due to dynamic metamorphism imposed at some time after intrusion and solidification, but that it is the texture induced at solidification in an intrusive under pressure and rendered unusually susceptible to schisting through the presence of an excess of mineralizers, especially water. The quartz veins have many resemblances to pegmatite dikes, and as such they must have a close connection genetically with the porphyry and in time relationship with the final differential and solidification of the mass. The fact that some porphyry masses are not connected with gold-bearing quartz is not adverse to the general theory, since the excess of water which seems to characterize the porphyry, where gold values are found, was probably a very local condition, confined to only a few masses and possibly there only to the advancing edges.

There are a number of minerals of diagnostic character associated with the Porcupine deposits and these present the most striking differences between the Dome and the Hollinger-

McIntyre areas. The Dome ores are noted for the presence of pyrrhotite, and recently gold tellurides have been discovered. These minerals are suggestive of high temperature conditions, as described by Lindgren in other pre-Cambrian gold deposits. They have not been identified in the Hollinger-McIntyre area. Tourmaline is found in both areas. In the Pearl Lake section it is most abundant at the east end (and upper side) of the quartz porphyry mass. Amorphous carbon, locally called graphite, has been encountered in the McIntyre, Hollinger, Porcupine Crown, and Vipond mines. It is especially prominent along lines of faulting. The faults are later than vein formation but older than the Keweenawan dikes. In the McIntyre it has been observed to occur also as a dust disseminated in the secondary minerals which fill the interstices between the incipient pillows and also in veinlets (with carbonates) in fractured basalt.

Deep mining has revealed the general shapes of the porphyry intrusions, especially around Pearl lake. Some structural conditions have been studied in the older formations, notably the Keewatin-Temiskaming contacts at the Dome mines, the mapping of basalt and greenstone areas at the Hollinger, and the location of basalt flow contacts at the McIntyre. The correlation and extension of these researches seem to me to offer a fruitful field for work which may have important economic results. Much of my own work on the Keewatin has been based on a study of a series of less altered basalt flows which outcrop in Cooke township, near Ramore, Ont. In the Porcupine district I have observed good flow contacts north of South Porcupine, on the road to the Davidson mine, where the flow tops face south, and underground at the McIntyre mine, especially on the 1,500-ft. level and 1,875-ft. level station, where the tops face north. In both cases the strikes are E.N.E. and the dips practically vertical.

MR. W. E. SIMPSON: Mr. Hopkins' paper is the work of one who, with his colleague, Mr. Burrows, has made an extensive study of the mineral deposits of Northern Ontario, and it fulfils a most useful educational mission. A concise description is given of each of the chief gold mines and thus it is a handy compilation of information and it emphasizes the most import-

ant fact that, so far as exploration has proceeded, the value of the ores has not decreased with depth. This is contrary to the experience of all other countries and the fact that such is the case greatly enhances the potential value of Ontario as a gold producer.

In discussing geological conditions favourable for ore deposition, Mr. Hopkins has given the very information that is needed by the prospector as a guide for the discovery of other ore deposits where the factors are similar. The paper may also be expected to fulfil a most useful mission in the hands of the many mine examiners whom we hope to see visiting Northern Ontario this coming season.

I am glad to know that this paper will be published as a separate booklet by the Ontario Department of Mines, and I hope it will be made available for public distribution at an early date.

# MINING AND MILLING METHODS AT THE HOLLINGER MINE.

By The Staff.

Annual General Meeting, Ottawa, March, 1922.

The Hollinger Consolidated Gold Mining Company was incorporated in 1916, and was an amalgamation of the properties of the Hollinger Gold Mines, Ltd., Acme Gold Mines, Ltd., Millerton Gold Mines, Ltd., and Claim No. 3147 of the Canadian Mining and Finance Company. The holdings are 440 acres in Tisdale township. Work was commenced on this group in 1909, but in 1911 the entire surface was wiped out by fire.

Milling started on June 15th, 1912, and since that time, though hampered by a labour strike and the World War, good progress has been made, so that at present the daily milling rate averages four thousand tons.

Since the geology of the camp is to be described very fully at this meeting by Mr. P. E. Hopkins, of the Ontario Department of Mines, this feature will be omitted from the present paper.

## The Mine.

*Development.*—Figure 1 shows the ideal layout of develop-

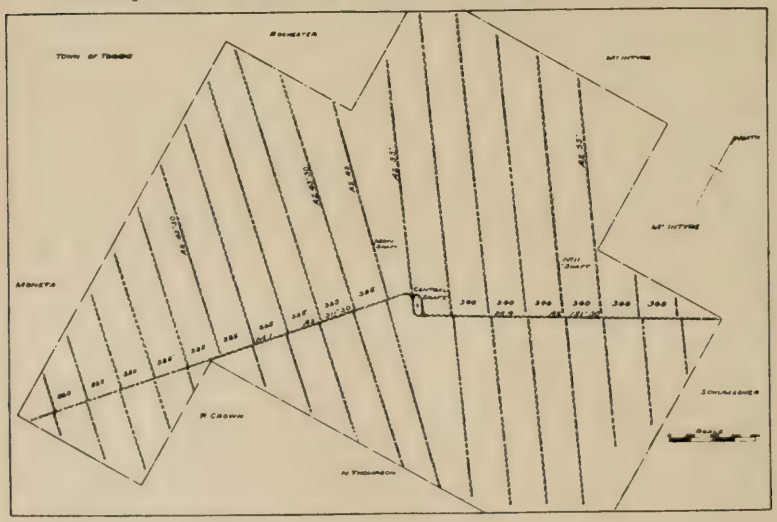


**Figure 1. — Ideal lay-out, lower levels.**

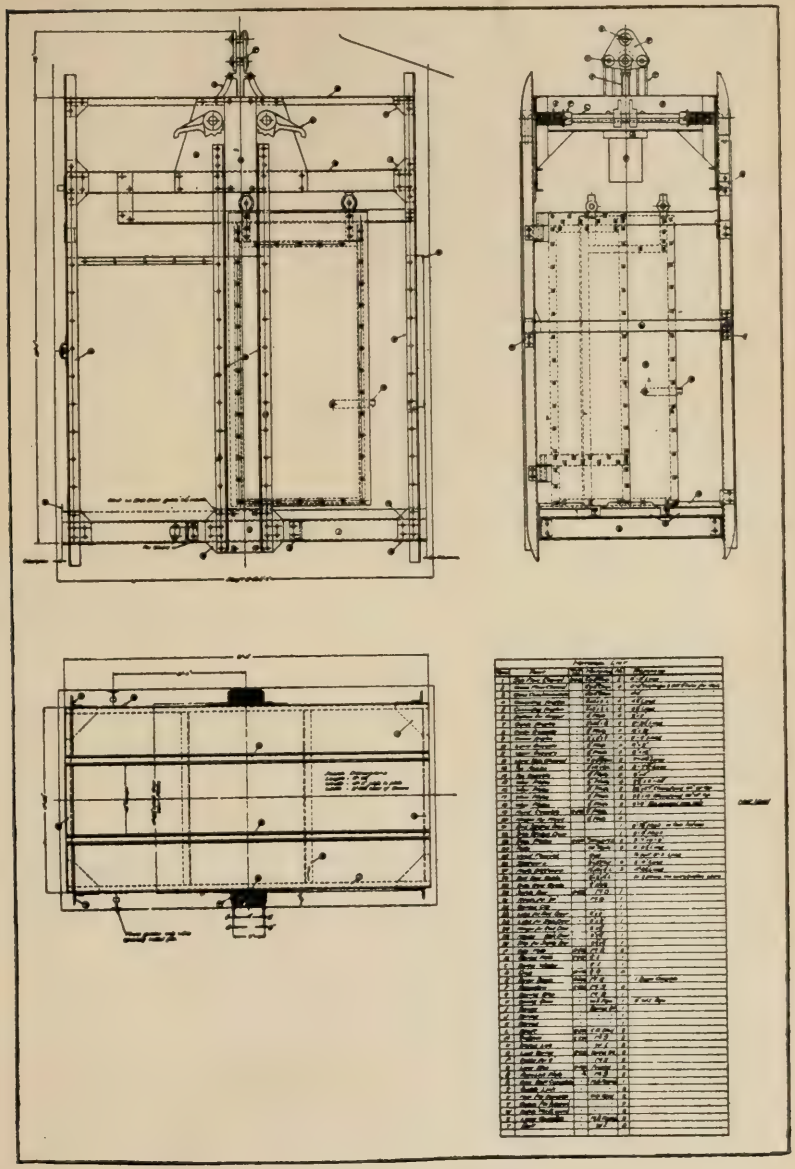


Figure 7. — Cage, No.11 shaft and Main shaft.

ment work. In practice all cross-cuts shown have not been extended as far as indicated. Three shafts will be carried to depth; the Main and No. 11 handling men and some supplies, such as steel, powder, etc., while Central handles ore, waste, and bulk supplies such as timber and rails.

The Main shaft has three compartments: one used as pipe and ladder way; one, equipped with single-deck cage, illustrated by figure 2, and carrying twenty men; and the third with double-deck cage, carrying 16 men. The double-deck cage-way of this shaft is being reserved for a separate ore hoisting compartment in case ore is encountered which will need different treatment from main flow. No. 11 shaft has a cage, similar to Main, and a pipe and ladder way. Central shaft is of six compartments. At present there are two skip-ways, two man cage-ways, pipe and ladder way, and one compartment for a material and service cage. The two man cage-ways will be turned over for skips for additional capacity.

The M-4 and M-1 series of cross-cuts are driven first on each level from Central shaft and connected to Main and No. 11. The other cross-cuts are then driven, turning off at each vein far enough for switches and protection for trolley boxes. All cross-cuts are driven 8 ft. 6 in. high by 7 ft. wide with 0.7% grade in favour of the load. Minimum curves are 30 ft. radius.

A development crew consists of four men — two drill off in the afternoon shift and two clean out drive in the morning shift. The round is blasted by a special blasting shift from 11 p.m. to 7 a.m. All development is contracted, and prices paid are $10.50 per foot for M-4 and M-1 drives, $9.00 for cross-cuts north, and $8.50 for cross-cuts south. This covers drilling, mucking, and explosives. Explosives are charged at a fixed price of 16 cents for 1¼ in. stick 50%, and 14 cents for 40%. Powder used on each drive by the blasting crew is posted each day so that the miners can check their expenses.

Drifting is also done on contract, at $9.00 per foot of advance of six feet wide, and $1.10 per ton for extra slashing. Veins are usually slashed to full width, then the back taken down to 17 ft. in height. This back work is contracted at 85 cents per ton for drilling, mucking, and explosives.

Drifts are timbered with stulls on 6 ft. centres. Formerly, local round spruce stulls were used, but lately 12 by 18 in. B. C. fir has been substituted, giving a considerable overall economy. Four-inch round lagging covers timber, and chutes are built every second set on 12 ft. centres. The chute is illustrated in Figure 3. The close spacing of chutes was

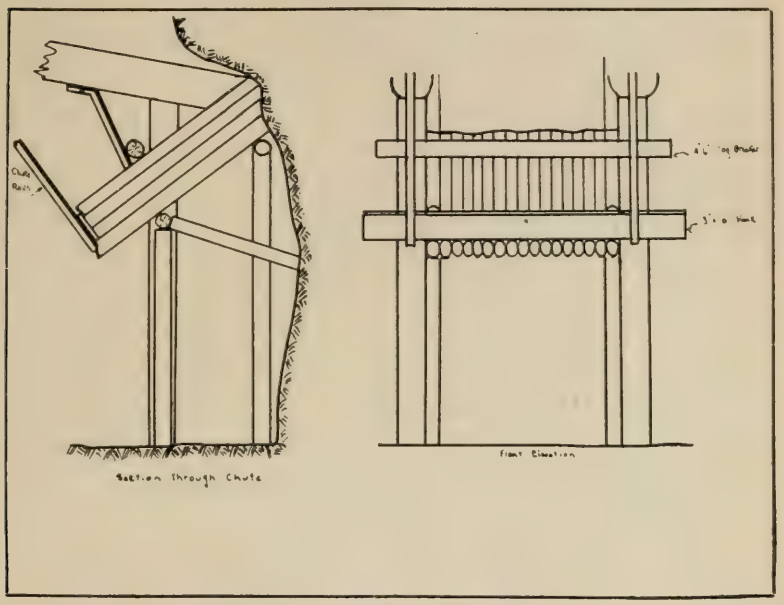


**Figure 3.—Standard stope chute.**

found to be economical — by assisting in robbing stopes, and also in the even drawing of muck to facilitate setting up in stopes.

*Diamond Drilling.* — Extensive diamond drilling is carried out, both as preliminary exploration and during production. Unfortunately the lenses are often anything but regular, and the drilling of the walls is of the utmost importance in obtaining the necessary information. All drilling is done by contract at prices from $1.10 to $0.80 per foot, depending on the nature of the rock and the percentage of core recovered. During 1921 the average paid was $1.04 per foot, and core recovery was 88%. Carbon consumption is 0.21 carats per 100 ft. All core is examined by the company's geologist every day, and the character of the rock and other particulars are plotted

with the assay results. Sludge samples are saved and used for check. Figure 4 illustrates a drill in action.

**Figure 4.—Diamond drilling.**

A modified air drill was devised to mount on a regular machine dump for side drilling stopes to from fifteen to twenty-five feet. This proved a handy and satisfactory machine for this work.

*Stoping.* — Since the ore is in more or less continuous lenses, standing almost vertical and with usually good walls and roof, flat back shrinkage stoping was adopted. Incline raises at 45 deg. are carried through to the level above, for use as pipe-ways, man-ways, and for ventilation. The irregularity of lenses necessitates some cribbed man-ways, and the occasional driving of raises partly in waste. Breasts are carried from 40 ft. to 50 ft. apart, and are usually 7 ft. high.

In a vein of 20 ft. width two rows of holes would be driven 8 ft. deep and 4 ft. apart. Where walls are irregular

and veins narrow, closer spacing and shorter holes are used. About one block holer is employed for each four machines. The average production in the stope is 27 tons per machine shift, average holes drilled 9.8 per shift, and powder used 1.3 lb. per ton. Machine men do their own scaling. Figures 5 and 6 illustrate stope scenes. "Bad hangs" in stopes are held by pillars. On completion of a stope to the level above, it is drawn. If walls are straight and few pillars left, no scaling is done; otherwise scalers follow the muck down, sweeping walls and pillars.

Figure 5.—Stope scene.

At present a modification is being used. From the 800 ft. level the stope is carried above the 675 ft. level, then drawn and filled. Then timber will be placed on the 675 ft. level and stoping carried on to the 550 ft., thence to the 425 ft. level. The 425, 800, and 1,550 ft. levels have been established as basement levels for the present, and here floor ribs will be left intact.

**Figure 6.—Stope scene.**

A system of sand filling is being devised, but it is not yet in operation, so no details are available.

*Underground Transportation.* — Figure 7 is a diagram of the ore and waste handling arrangement at Central shaft. Figure 13 shows gate at each level on each pass.

Above the 425 ft. level all rock is tipped into passes, drawn out on the 425 ft. level, and hauled to either ore or waste passes at Central shaft.

Figure 8 illustrates a loading chute and the 20-cu. ft. cars used. These ore passes are all connected as illustrated, and they give an available storage behind the skips of 6,000 tons above the 950 ft. level.

Below the 425 ft., all levels are equipped with trolley locomotives, and the rock is hauled direct to the shaft. Side tipping cars of 20- and 26-cu. ft. capacity are used.

All cross-cuts are equipped with 35lb. rails, with ties spaced at 2 ft. centres. Ties are 4 in. thick and are made by sawing

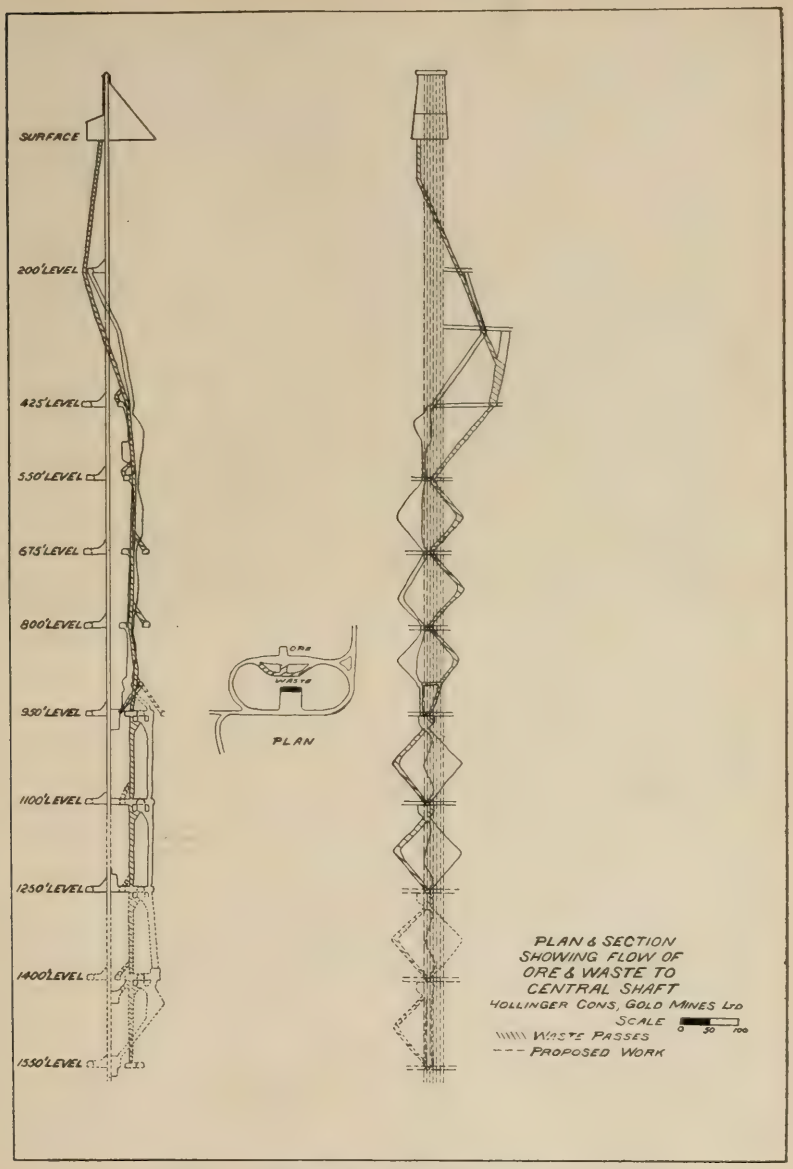



**Figure 7.—Plan and sections showing flow of ore and waste. Central shaft.**

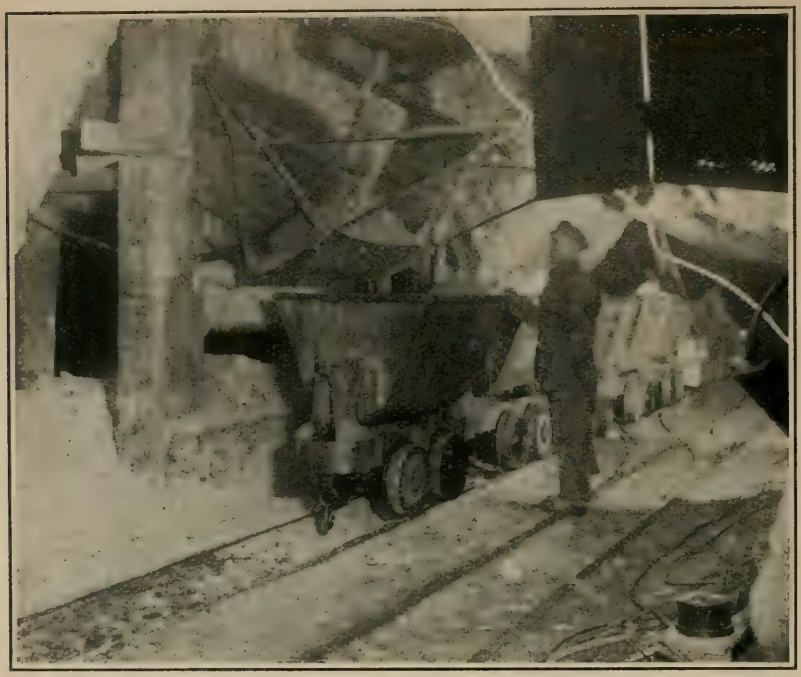


**Figure 8. — Loading chute and 20 cu. ft. cars.**

flats on the larger pieces of spruce lagging. Drifts have 20 lb. rails on similar ties.

Except for the two storage battery locomotives, which are used on development, all locomotives are of the trolley type, eight being in service. These are 18 in. gauge, two motor type, with 1,500 lb. draw-bar pull and a speed of 6 miles per hour. They haul trains of 50 cars to the dump, but the average number of cars is 25. This locomotive is illustrated in Figure 9.

On the levels below the 800 ft., the gauge is being increased to 36 in., with three-ton cars, which will be dumped on tipples.

Power is supplied by a 250 Volt, 50 K.W., D.C. motor-generator set at the shaft collar, and a similar one on the 800 ft. level.

*Tramming and Loading.* — From the loading chute,

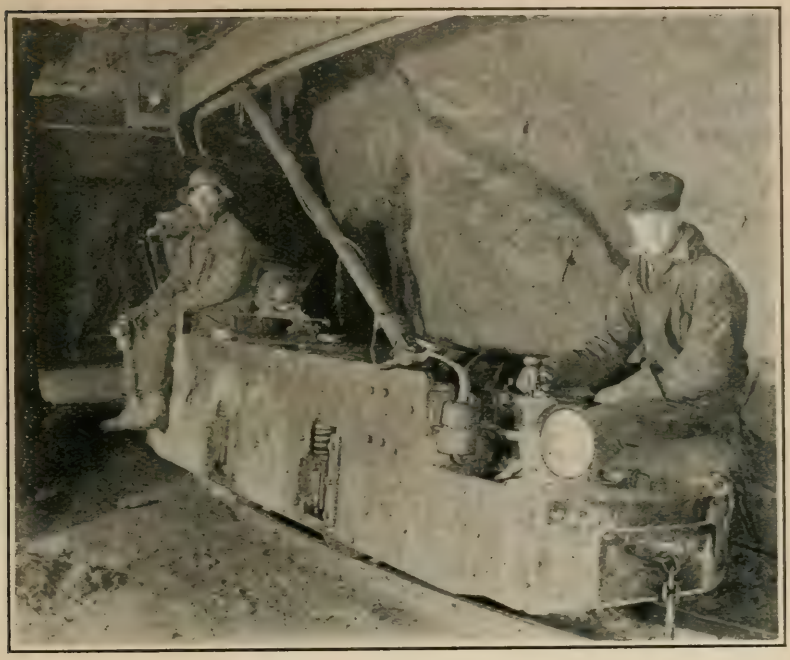


**Figure 9. — Mine locomotive.**

illustrated in Figure 8, a crew of three men draw 450 cars as a shift's work. From the stope chutes they range from 20 to 50, and average 32 tons per man shift. The ore often breaks in slabby shapes which gives considerable trouble to the chute men.

*Sampling.* — Extensive sampling is necessary for close checking of operation, as well as for estimation of ore reserves. All cross-cuts are sampled on both sides, and every development face is sampled daily.

In stopes, 80% of the breasts are sampled, less attention being paid to stopes where the ore is comparatively uniform. Special care is given to sampling pillars, raises, and the last cut under the floor pillar. Test holes (either by standard diamond drills, Leyner type hammer drills, or by the small diamond drill previously referred to) are very extensively employed to give information regarding walls. Samples

average 4 in. wide and yield 1 lb. per foot. Samplers take ten 4-ft. samples in cross-cut per shift, but fewer in stopes. About 300 samples are turned in daily. Figure 10 illustrates a stope sample cut.

Figure 10.—Stope sampling scene.

A new departure is the use of electric furnaces for general assaying. At first, one was used experimentally for assaying gold solutions, and the highly satisfactory results obtained determined the installation of six additional and larger electric units, which are adaptations of the heat treating furnaces used in the automobile industry. These are proving very satisfactory.

*Blasting.* — Blasting is practically all done on the 11 p.m. to 7 a.m. shift by a special crew. Powder is sent to a central thawing magazine on the 550 ft. level every morning, when it is racked and thawed at a temperature of 68 to 70 deg. F., this temperature being maintained by air circulation through an electric heater. Blasting bosses get the detail of work required from afternoon shift bosses.

Blasters go in pairs, and each team blasts an average of 80 holes in stopes and development faces. In stopes, 1.3 lb. of 40% Forcite is used per ton, and in drifts, 2.26 lb. per ton.

The equivalent of 16.46 lb. of 50% Forcite is used per foot of cross-cut.

*Steel.* — One and one-eighth inch hexagon hollow steel is used almost exclusively, although some 1⅛ in. octagon is still in service for the small amount of dry stoper work now done. All steel is X-bit, with straight 14 deg. taper and cutting edges 90%. The run is 14 in., with ⅛ in. change.

In sharpening 1¼ in. steel of 14 deg. pitch, round, wet dollies are used in pneumatic sharpening machines.

Steel are heated in oil-burning furnaces of local design, made to take steel both sides. High pressure burners are used, which consume 16 gallons of kerosene per shift. An average of 50 steel are sharpened per gallon of oil consumed.

A steel sharpener and helper will dress and temper an average of four hundred steel per shift of eight hours. During 1921 one steel was sharpened for each ton of rock broken.

*Air and Water.* — The compressor plant is equipped with two Corliss valve 4,500 cu. ft. fixed capacity compressors, one cam-operated valve, variable capacity 4,500 cu. ft. machine, and one spring valve 9,100 cu. ft. machine set to unload 50% to 100%. All compressors are driven by synchronous speed, direct connected, 12,000 volt motors, the large one being 1,232 K.V.A., the others 600 K.V.A., 24 pole, 125 R.P.M.

A unique feature of the installation is the availability of the two constant speed compressors as steam engines in case of a power shortage. A steam governor is connected up and steam introduced to the air end, when the motor becomes a generator. This proved a valuable asset during the recent power shortage. Steam for the above and for fire pumps is furnished by a battery of four vertical tube water boilers of 300 H.P. each.

Water, for rock drills and drinking, is purchased from the town of Timmins, and, together with air lines, is taken down Central shaft and distributed along each level through pipes hung on trolley wire supports. Mine pumping at the 425 ft. level is done by a 10 by 12 in. Triplex pump, which discharges into an 8 in. pipe line through Central shaft to the mill. At

the 800 ft. level a 7½ by 10 in. Triplex is located, and pumps to sumps at the 425 ft. level.

*Hoisting.* — All rock hoisting is done at Central shaft by a double cylindrical drum electric hoist. Drums are 10 ft. in diameter.

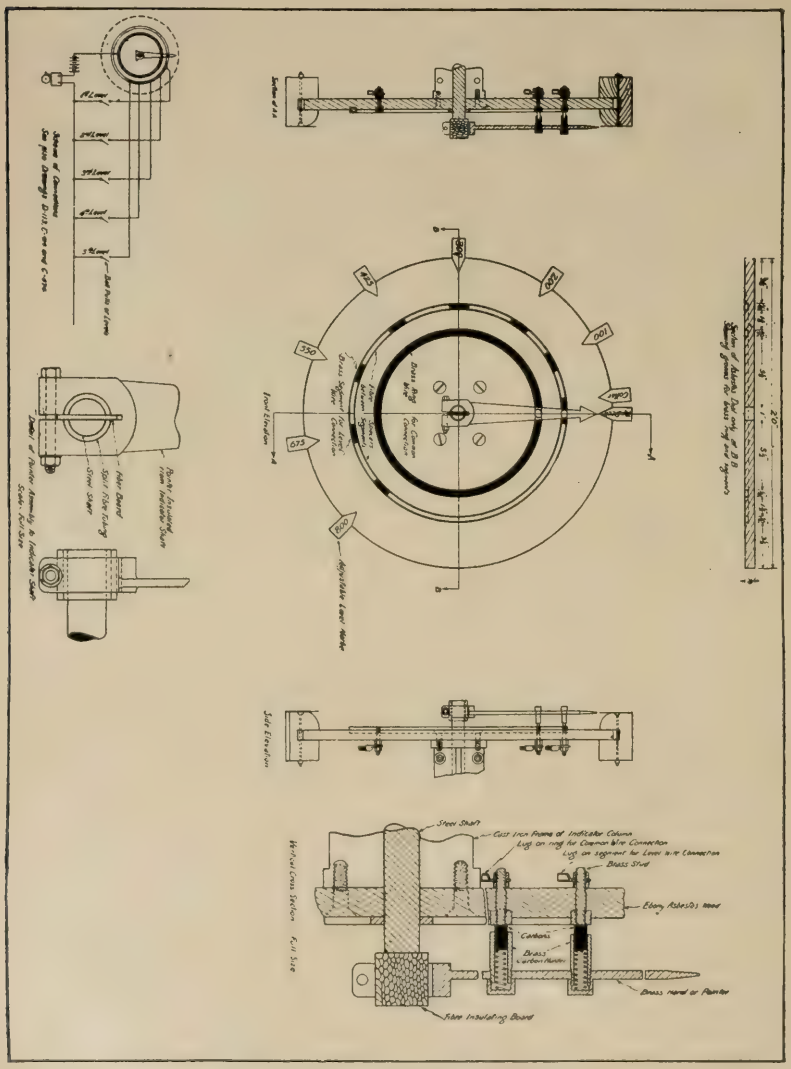



Figure 11.— Depth indicator with electrical contacts

The hoisting speed is 1,200 ft. per minute, with 500 H.P. slip ring motor, using rheostatic control. Two five-ton Kimberley skips are operating in balance, and hoist a maximum of 5,200 tons of ore per day. Waste is hoisted in the same skips at present, and tipped into a waste bin, whence it travels through a system of passes to the main waste system, where it can be taken off for filling at any level from 425 ft. down.

The cable used is 1⅜ in. diameter, 6 strand, 19 wire, Lang's lay, plow steel. The last one, which was taken off in August, ran 523 days, and hoisted 559,000 tons. This hoist is equipped with a Johnson recorder, which is a most important aid in localizing trouble.

Men are hoisted at Central shaft on double deck cages in balance. Hoist is double drum, cylindrical, electrical, with Ward-Leonard control. At Main shaft, a geared electric is used, and at No. 11 a temporary air hoist. A feature of the hoisting signal system is illustrated by Figure 11. The pointer is geared as usual to the hoist shaft, and contact can only be made in the bell circuit when the cage is at one or other of the levels — an important safeguard.

## THE MILL.

Milling of the ore consists of four steps:

(1) Reduction of the ore to such a fineness that its precious metal content will be exposed to the action of the extracting solutions;

(2) Extraction of the values by treatment with sodium cyanide solutions;

(3) Separation of the solution containing the values from the solids from which the valuable content has been extracted; and

(4) The recovery of the values from the solution by precipitation, and the refining of the precipitate.

Reduction of the ore is divided into *Coarse Crushing* and *Fine Grinding.*

*Coarse crushing* is done with three No. 7½ gyratory crushers, four No. 5 gyratory crushers, and one set of 40 by 20 in. rolls.

The ore, having passed, while underground, through a 14-in. grizzly, is dumped from the skips over grizzly bars, the oversize going to the two outside No. 7½ crushers and the undersize to the centre one. The grizzly bars are 8.5 feet long, rest at an inclination of 35 deg., and have an opening

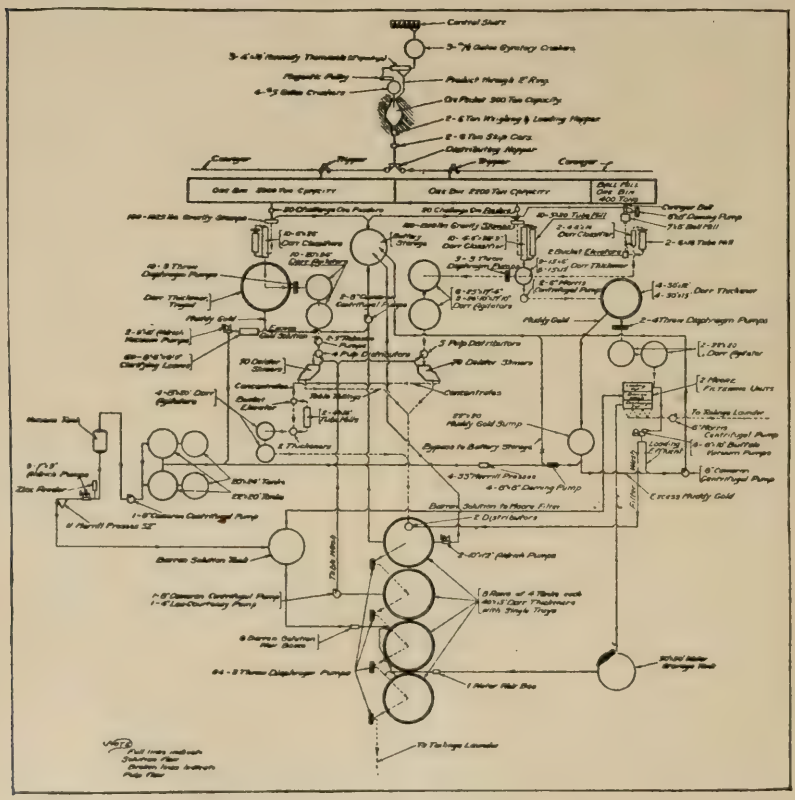


**Figure 12.— Flow-sheet in mill**

3½ inches wide at the top and 7½ inches at the bottom. Greater capacity is obtained by separating the fines from the coarse, especially in handling wet ore.

Each No. 7½ crusher, set to give a 3½ inch product, has an eccentric speed of 144 R.P.M., and requires 50 H.P.

for its operation. The capacity of each crusher is 70 tons per hour, with a maximum of 90 tons per hour. Upper concaves are of white iron and have a life of 12 months, while the lower are of manganese steel with a life of 8 months. Concaves are re-set every 4 months and take, on an average, 1,700 lb. of zinc per setting, 80% of which is recovered.

Eccentrics are re-babbitted every 4 months and take 90 lb. of babbit. New high-grade babbit is used. Head centres are of cast steel. Mantles are of manganese steel and are drawn on to the shaft under pressure of 125 tons, after warming to 120 deg. F. Poured zinc keys prevent the mantle and head centre from turning on the shaft. With the head centre giving full length support to the mantle throughout, the life of the mantle has been prolonged 30%. Mantles have a life of 7 months.

Four 4 by 16 ft. revolving screens, taking the discharge from the No. 7½ crushers, have 2 in. diameter round openings, make 15 R.P.M., and have an inclination of 1 11/16 inches to the foot. Screen plates are of manganese steel 1⅝ in. thick, and have a life of 12 months. Trommels are unnecessarily long for their present work.

Fines from the trommels drop through chutes to the crushed ore pocket, while the oversize discharges over a magnetic pulley to the No. 5 crushers. The oversize from each trommel will run to two No. 5 crushers.

The magnet pulley merits special mention. The pulley is 24 inches in diameter with a 29-inch face. The pole pieces, five in number, are spaced on 6-inch centres, and are one inch wide. Brass rings are used as distance pieces and are covered with rubber belt to take the wear. Each pole piece has an iron ring, cast in sections, fastened to it with cap screws with counter sunk heads. Thus the poles of the magnet extend one inch beyond the face of the pulley. The magnet is powerful enough to hold any steel that comes up in the ore, and will attract and pick up any small pieces that would not otherwise come in contact with it. The steel clinging to the pulley trips out a switch as it passes around the pulley, thus breaking

the circuit to the magnet, which thereupon drops the steel behind the ore streams. The moment the circuit is broken, the attention of the operator is called by the turning on of a light. The switch is closed by hand. Though its operation is not entirely automatic, the efficiency of the magnet is 100%.

The No. 5 crushers are driven individually from 25 H.P. motors and have an eccentric speed of 157 R.P.M. For each crusher 25 H.P. is required. Head centres, mantles, and concaves are of the same material and design as those in use on the No. 7½ crushers. The life of the upper concaves is 12 months, and of the lower concaves 3 months. Concaves are re-set every 4 or 5 weeks and require 900 lb. of zinc. Eccentrics are re-babbitted every 12 weeks and take 85 lb. of babbit.

Two crushers, set to 1½ inch, and one of which is usually standing as a spare, discharge to the crushed ore pocket. The remaining two crushers, set to 2½ inches, discharge on

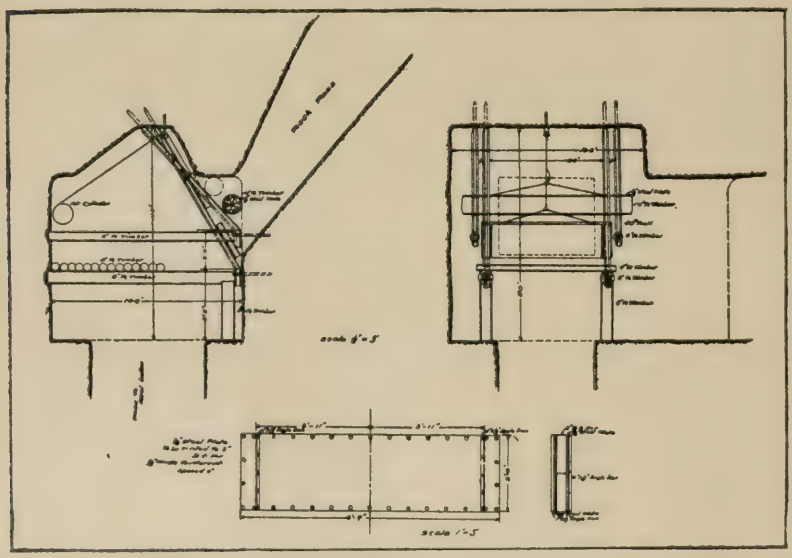


Figure 13.— Permanent chute, ore and waste passes.

to a 24-inch conveyor belt. The crushers set to 1½ inches cost twice as much for upkeep as those set to 2½ inches. Chutes are so arranged that the discharge of any crusher may

be sent either to the rolls or to the crushed ore pocket. The 24-inch conveyor, running 200 feet per minute, delivers the ore to the rolls.

The rolls, erected as an experimental unit, but at present operated as an integral part of the crusher plant, are 40 inches in diameter, 20-inch face, with corrugated manganese steel shells four inches thick. Power is supplied directly from a 24-inch diameter pulley on a 480 R.P.M. motor to a 108-inch diameter pulley on the fixed roll shaft. A double throw switch allows the motor to be reversed to unload the rolls in case of a block. Finger gears keep the corrugations in mesh. Rolls make 110 R.P.M., giving a shell speed of 1,150 feet per minute. One set of roll shells, weighing 4,730 lb. and set to three-quarter inch opening, crushed over 200,000 tons of ore, giving a steel consumption of less than .024 lb. per ton of ore crushed. Rolls take 45 H.P. and have a capacity of 125 tons per hour.

Ore is fed into the rolls from a shaking feeder. The upper housing of the rolls is cut down to allow the lip of the feeder to come as low as possible and directly over the opening between the rolls. With a properly distributed and thoroughly mixed feed the life of the shells will be materially increased.

A 24-inch inclined belt conveyor running 295 feet per minute returns the roll discharge to the crushed ore pocket. The crushed ore pocket, 80 feet deep, is underground, directly below the crusher house. It has an effective capacity of 750 to 800 tons.

A 4-ft. diameter fan extracts the dust from the crusher house through a suction pipe connected to the top of the crushed ore pocket.

The crusher plant is operated 19 shifts per week, shaft inspection and skip and crusher repairs being made on the other two shifts. The operating crew consists of crusher man and four helpers, one on rolls, one on the No. 5 crusher, and two on the No. 7½ crushers.

*Conveying.* — Ore is conveyed from the bottom of the crushed ore pocket to the mill in 6-ton skip cars, operated from a hoist in the mill, 900 feet distant. Skips are filled from 6-ton weighing hoppers suspended from scales. Roll feeders, driven by cams operating lever arms which grip friction wheels keyed to the roll feeder shafts, deliver the ore from chutes at the bottom of the ore pocket into the weighing hoppers.

Solenoids in an electric circuit are used to stop and start the feeder. The rising of the scale beam breaks the circuit, allowing a dog to drop under the arm holding it clear of the cam. When the circuit is closed, the dog is drawn out and the feeder is again in operation. A pawl locks the lever for dumping the hopper when the circuit is closed, *i.e.*, while the hopper is being filled. The circuit is also broken when the hopper door is open, so that the feeder will not operate till the door is closed. Hoppers are balanced every two hours and allowance is made for anything sticking to them.

The skip cars, two in number, run on two standard gauge tracks to the pent house on the top of the mill. Cars are hauled by a ¾-inch cable winding on 66 by 70 in. drums on the hoist geared to a 100 H.P. motor. Crucible steel cable of 6 strands, 19 wires to the strand, is used. Cars run 700 feet per minute, making the one-way trip in 85 seconds, including time for loading and dumping. A maximum of 46 skips per hour can be hauled. One cable will haul 300,000 tons. Out-riding wheels on the front axle of the skip cars run up a curved track rivetted to the sides of the dumping plate, lifting the front of the car sufficiently to bring the bottom of the car to an angle of 45 deg. from the horizontal. The ore runs out over the slanting back of the car into the hopper below.

A tail rope of discarded haulage rope, or of ½-inch wire cable, is used, armoured with rubber hose and sections of split pipe where ore is dumped over it. The tail cable at the lower terminal turns around a 27-inch diameter sheave on a vertical shaft mounted on a weighted car free to move on an inclined track. This allows for sag and stretch of cables. Cables are supported on wooden rollers, belt covered, at intervals of 30

feet along the track. Guard rails over the regular track prevent a car from leaving either terminal unless squarely on the track. Dogs held in position by shearing pins lock the skip in the dumping plates in case of an over-wind.

The hopper below the dumping plates will hold 25 tons. Chutes with roll feeders at the bottom lead from the centre of the hopper to the conveyor belts, of which there are two. No. 1 conveyor feeds the ball mill and the first 100 stamps, called the No. 1 mill. No. 2 conveyor feeds the second 100 stamps, or No. 2 mill. The conveyors are 24 inches wide of 6-ply belt, with 3/16 in. rubber face and edges, and 1/16 in. rubber back; No. 1 operates at a speed of 396 feet per minute, and No. 2 at 308 feet. Belts have a life of from 30 to 36 months. Each belt is driven from a 25 H.P. motor which may be tripped out in an emergency from any place along the conveyor belt. One man operates the conveyors, of which only one is in use at any time. Ore is distributed by means of a hand operated conveyor tripper. The hopper is emptied in 50 seconds after the car is dumped.

Ore bins have a capacity of 5,600 tons. Shovellers have to be put in the bins in the case of more than 8 hours' delay in tramming, as the ore will stand up at an angle of 70 deg. from the horizontal.

*Fine Grinding.* — The ore is reduced through ⅜ in. mesh screens in a ball mill and 200 stamps, and finished to 70% through a 200-mesh screen in 22 tube mills.

The ball mill is 7 by 6 ft. inside the shell, and is lined with manganese steel liners. The shell liners are stepped from 2 inches to 4 inches in thickness. End liners are thickened at the place of greatest wear to 3 ¼ inches. A complete set of liners weighs 20,000 lb. and has a life of 8 months. A discharge screen, with ⅜ by 1 15/16 in. openings over an area 24 inches in diameter, is used. The screen is of cast steel and lasts 6 months. The ball load is kept at centre line of the mill, and weighs between 11 and 12 tons. Four six-inch diameter chrome steel balls are fed per shift, making 396 lb. per day. The ball mill is run at 25 R.P.M., and takes 145 H.P.

The feed has 20% moisture, and is fed into the mill from a conveyor belt, dropping the ore into the 45 deg. connection of an 8-inch Y-branch, of which the long axis is in line with the axis of the mill. A 2-inch all-iron Pelton nozzle delivers the solution at a pressure of 80 lb. into the centre of the outer end of the Y-branch. The descending ore is struck by the stream from the nozzle, and is hydrauliced through an 8 by 6 in. nipple into the mill through the open trunion bearing. A curved white iron liner, fitting into the bottom of the Y-branch and nipple, extends 2 inches inside the trunion bearing. A small tailing wheel bolted to the trunion face plate catches the drip and, carrying it up, drops it into the feed stream through an opening in the top of the nipple. Pressure for the nozzle is supplied by an 8 by 8 in. three-throw plunger pump, belted to the ball mill pinion shaft. Pressure is controlled by regulation of the by-pass on the pump. Balls are fed to the mill by dropping them into the feed hopper.

A conical screen with $\frac{3}{8}$ by $\frac{5}{8}$ in. openings is fastened to the discharge of the mill. The oversize — 7 or 8 tons per day — is returned by a 6-inch belt and bucket elevator to the ball mill feed. The ball mill is followed by two 6 by 16 ft. tube mills in closed circuit with two 4 ft. 6 in. by 16 ft. Rake duplex classifiers. The ball mill and tube mill discharge is returned to the classifiers by a 24-inch belt and bucket elevator. The head pulley is 36 inches in diameter and is run at 47 R.P.M. The classifier sand discharge is conveyed to tube mills by two 10-inch screw conveyors, running at 48 R.P.M.

The tube mills have white iron shell liners and white iron end liners. Discharge screens are of cast steel, with $\frac{3}{8}$ by $\frac{3}{4}$ inch openings. Shell liners last 22 months, end liners 16 months, and screens 6 months. The tube mills carry a pebble load of 11 tons, and are kept at a level 5 inches above the centre line of the mill; 800 lb. of 4-inch flint pebbles are fed to each mill per 24 hours. Each tube mill, making 24 R.P.M. and taking 115 H.P., is driven with a spur gear and pinion connected to the motor through an enclosed reduction gear set.

Classifier rakes make $15\frac{1}{2}$ strokes per minute. The ball mill and two tubes reduce 600 tons per day to 70% through

200-mesh. The circulating load of each tube is approximately 600 tons per day. Ball mill feed is 70% through 1-inch ring, and 45% through ½-inch ring. Ball mill discharge has 15% through 200-mesh. The ball mill circuit is operated by one man.

*No. 1 Mill Stamps and Tubes.* — No. 1 mill stamps and tubes are divided into 10 units, each composed of ten stamps, one tube, and one classifier, driven from one 125 H.P. motor.

Stamps with new shoes weigh 1,500 lb. and drop 99 times per minute through a distance of 7½ inches. Screens having ⅜ by ⅝ in. or ¼ by ½ in. openings are used, with a 2-inch height of discharge. All wearing parts other than mortar boxes are of chrome steel. Split steel stem guide boxes with bronze bushings are used. The stamps are driven from a counter shaft driven with a bevel gear from the tube mill pinion shaft. The gear on the counter shaft has teeth of hard maple, fitted and finished in the mine shops. A 48-hour treatment with boiled oil after finishing has increased the life of the teeth from 4 months to 14 months. Ten stamps require 35 H.P. for their operation. Cam-shafts are 7½ inches in diameter, 8 ft. 8 in. long, turned down to 6½ inches at the bearings, and they carry 5 cams each. Bull wheel flanges of cast steel are pressed on to the shaft with a hydraulic jack. No flange so put on has come loose since this practice was adopted six months ago. Two cast steel collars, three inches thick, are shrunk on the cam-shaft, one at each side of the outer bearing, to prevent end play.

The tube mills are 5 by 20 ft., with the same style of drive and linings as the 6 by 16 ft. mill of the ball mill circuit. Tube mills make 28½ R.P.M., carry 9 to 9½ tons of pebbles, and are kept 5 inches above the centre line of the mill; 800 lb. of 4-inch flint pebbles are fed to each mill per day. The tube mill and classifier requires 75 H.P. Each unit reduces 163 tons a day to 70% through 200-mesh. The feed to the stamps is the same as to the ball mill. The circulating tube mill load is approximately 350 tons.

The Duplex classifiers are 4 ft. 6 in. wide, lengthened out to 29 feet. This allows for a 2 inch to the foot grade from

tube mill discharge to the classifiers, and a grade of 4 inches to the foot for the sand discharge of the classifier to the tube mill feed scoop. Classifiers are set on a grade of 2¼ inches to the foot, and have a pool 18 inches deep at the overflow, giving a settling area 4 ft. 6 in. by 8 ft. Classifier rakes make 17½ strokes per minute.

*No. 2 Mill Stamps and Tubes.* — No. 2 mill stamps and tubes consist of 10 units, having ten stamps, one tube, and one classifier to each unit. These stamps, each of which weighs 1,625 lb. and drops 105 times a minute through a distance of 7 inches, require 40 H.P. for their operation. Screens and height of discharge are the same as in No. 1 mill. Cyanide solution is fed with the ore to the batteries in the ratio of 5 to 1. The tube mills are identical with those of the ball mill circuit, except that they are operated at 28 R.P.M. The tube mill pinion shaft is belt driven from a 100 H.P. motor. One tube and classifier take 97 H.P.

The Duplex classifiers are 6 by 26 ft., set 2.5 inches to the foot. Rakes make 17.8 strokes per minute. Classifier feed and sand discharge launders are set at a lower grade than in No. 1 mill and cause some trouble from being too flat. The capacity of each unit is 186 tons per day. The circulating load of the tube mill is 470 tons per day, approximately. All tube mills are operated with 31% moisture in the pulp, gravity being controlled by the addition of solution to the sand discharge of the classifiers. Gravities of tube discharges are measured and recorded every 2 hours.

A crew of 7 men operates the batteries and makes all repairs other than changing broken cam-shafts, stems, tappets and cams. Three men operate the tube mills. A battery shift boss has the oversight of the conveyors, ball mill, stamps, and tube mills.

*Cyanidation.* — The classifier overflow from No. 1 mill classifiers flows to fifteen 13-ft. diameter thickeners, which give a muddy overflow. The pulp, thickened to from 50 to 45% moisture, is drawn off from the bottom of the thickeners with five diaphragm pumps into six 25 ft. by 17 ft. 4 in., and

two 26 ft. 10 in. by 17 ft. 10 in. agitators. These agitators are run at 8½ R.P.M., and require 6 H.P. each. The muddy overflow from the 13-ft. thickeners, composed of 880 tons of solids with 95% through 200-mesh, and 9,500 tons of solution, is pumped with two 6-inch centrifugal pumps to a tank distributor, whence it is fed to eight 30-ft. diameter thickeners. The pulp, thickened to from 50 to 60% moisture, is given two-step agitation in two 22-ft. diameter 20-ft. deep agitators. The discharge of the agitators flows to the leaf filter loading vats. The filter plant has two units with the usual 3-vat arrangement. There are two baskets of sixty 6 ft. 4 in. by 6 ft. 10 in. leaves to each unit. A 35-ton electric crane operates both units. The average cycle is: 15 minutes loading, 30 minutes barren wash, 10 minutes water wash, and 45 minutes to discharge. The average load per basket is 18 tons of solids. Cake is reduced to from 28 to 30% moisture. With solution carrying $4.00 in gold, and 1 lb. of cyanide per ton going to the filter, the solution loss per ton of ore is less than one cent. Filter discharge, diluted to 70% water, is pumped with a 6-inch centrifugal pump into the tailing launder. The high grade effluent from the filters is returned to the battery storage, while the washes are used to dilute the feed to the decantation plant.

The clear overflow from the 30-foot thickeners runs to the muddy gold sump. Triplex plunger pumps, 8 by 8 in., making 50 R.P.M., pump from this sump to four 34-inch 20-leaf pressure clarifiers. Each clarifier has a capacity of 45 tons per hour. The excess solution from the muddy gold sump is returned to battery storage.

In No. 2 mill the classifier overflow is thickened to 50% solids in five 40 by 14 ft. trayed thickeners. The thickened pulp is pumped with ten diaphragm pumps to ten 20 by 24 ft. agitators operated at 5.4 R.P.M. and taking 2½ H.P. each. The pulp is given two-step agitation, and flows through a chip screen to the feed box of two 5-inch centrifugal pumps.

The chip screen is 3 ft. wide by 6 ft. long, set at a grade of 1.5 inches to the foot. It is mounted on the underconstruction of a concentrating table, and is operated with

the usual head motion. The pulp flows through the screen punched with holes 1/8 in. in diameter, while the chips are projected over the tail into a box, after having been given a wash of barren solution. The screen has a capacity of 2,000 tons solids per day.

The centrifugal pumps elevate the pulp to the table distributors of No. 2 mill. The agitator discharge of No. 1 mill runs by gravity to the table distributors.

Concentration is performed on 25 double-deck and 88 single-deck slime tables, giving an average tonnage of 23 tons per deck per day. The concentration ratio is 10 to 1. The table tail either flows or is pumped to the decantation plant.

The table concentrate is conveyed by drag belt conveyors to the boot of a 24-inch belt and bucket elevator, discharging into two 13-ft. diameter thickener classifiers. The thickened pulp is sent to two 4 by 16 ft. tube mills, making 34 R.P.M., and filled 2 inches above the centre line with 1¼-inch balls. The tube mills are lined with white iron wavy liners, 3 inches thick, which have a life of 6 months. The ball load is 12 tons per mill, and 700 lb. of balls are fed to each mill per day. Each tube mill takes 75 H.P. Tube mill feed is 20% moisture. The concentrate classifier overflow pulp has 85% through 200-mesh, which after agitation has a value of 45 cents per ton. Classifier overflow is given two-step agitation in four 15 by 20 ft. agitators. The agitator discharge flows to the decantation plant.

The clear overflow from the 40-ft. thickeners runs in launders to two clarifying vats, 18 ft. 7 in. by 9 ft. 4 in., with space for 100 leaves. The latter, 8 ft. 4 in. by 6 ft. 6 in., are connected with ¾-in. pipe and fittings through a 10-inch header to two 9 by 10 ft. vertical, valveless suction, wet vacuum, Triplex pumps. Each leaf has a capacity of 3 tons per hour, with a 16 to 20 in. vacuum. The leaves are washed every second day with a hose, and are treated with 1½% HCl solution every two weeks. Sixty to sixty-five leaves are used, and a 20-inch vacuum is maintained. The canvass used is No. 10 duck.

*The Decantation Plant.* — The decantation plant consists of 8 rows, with four tanks to each row. Tanks are 40 feet in diameter by 14 feet deep, and are equipped with steel trays. The clear overflow from the bottom compartment rises through 6-inch pipes inside the tank, and joins the overflow from the top section. The flow from the bottom compartment is controlled by regulating the head of the discharge pipe with 6-inch iron rings.

A specially designed launder ensures giving the same class of feed to both compartments. An annular launder, 7½ in. wide, 24 in. inside diameter, and 6 in. deep, is suspended from two ½ by 3 in. bars clamped to the thickener shaft. The launder rests on two L-shaped hangers, with holes in the vertical pieces which pass between the two bars clamped to the shaft. Two bolts through the hangers suspend the launder from the clamps. Two 6-inch pipes extend from the bottom of this annular launder, down into the boot of the lower compartment. The feed launder with a splitter is made to discharge half into the annular launder, which feeds the bottom compartment, and half inside the launder, where the spider spreads it out over the top compartment.

Slime levels are measured and recorded every two hours. The slime level in the lower compartment is measured by means of an adjustable syphon, which is lowered in a pipe through a hole in the tray till it runs muddy. The slime level is read from a scale placed at the back of the adjustable leg of the syphon.

The thickened pulp is pumped with a three-pot diaphragm pump connected to each tank compartment. A three-step pulley allows for three speeds which, with the removal of discharge valves, give a range of 9 capacities to the pump. All valves and valve seats are of rubber. Diaphragms have a life of 150 days. One pump takes about ½ H.P. Each thickener makes 1 revolution in 4¾ minutes, and takes ⅜ H.P

The operation is the standard counter-current continuous decantation practice. The overflow from No. 1 tank is returned to battery feed storage. Overflow of No. 2 tank is used for table wash and table feed dilution. No. 3 overflow dilutes the

feed to No. 2 tank, while No. 4 tank overflow dilutes the feed to No. 3 tank. Barren solution is added to No. 3 tank, and water to No. 4 tank.

Gravities of discharge vary from 1.47 to 1.64, depending on the percentage of slow settling material in the ore. To obtain satisfactory settling, lime must be kept to at least 0.45 lb. of CaO per ton of solution in No. 4 tank. Temperatures below 68 deg. F. also reduce the settling rate. Steam is being used during cold weather to heat the water fed to No. 4 tank, but results are not yet conclusive as to whether or not it is of commercial value. The only heat being sent to the mill is that added to the water going to No. 4 tank. The loss of gold in solution from the decantation plant for the year 1921 was 8 cents per ton of ore milled. The decantation plant is operated by three men per shift.

One 14-ft. diameter by 16-ft. drum type filter was installed in August 1921. Four months' operation have demonstrated that a net saving of 7 cents per ton may be made by filtering the decantation plant tails. The saving in cyanide will pay for the operation of the filters.

*Precipitation and Refining.* — The clarified gold solution is collected in two units of two tanks each. One pair of tanks is filled while the other is being precipitated. A tell-tale, operated by a float, indicates the tank tonnages at all times. By recording the readings before each switch is made, an accurate record of the tonnage precipitated is obtained. Vacuum treatment in conjunction with zinc dust precipitation is used. After presses are cut in and properly primed, .05 lb. of zinc dust is fed per ton of solution precipitated. The solution is pumped to the precipitate presses by 7 by 9 in. Triplex plunger pumps, making 50 R.P.M.

The gold precipitate is filtered through eleven 54-inch triangular presses of 20 leaves each. Frames are dressed with No. 10 duck and covered with filter paper. Each press has a capacity of a little less than 700 tons per day for a 7 day period under present operating conditions. Presses are cleaned every 7 days, or oftener if pressure goes above 30 lb. per sq. in. The filter paper is removed with the precipitate. The duck has a life of 12 weeks.

The precipitate is dropped from the presses into cars on a track below and trammed to the refinery. It is fluxed with borax, litharge, pyrite cinders, and quartz, and smelted in a lead blast furnace, producing a base lead bullion 180 to 200 fine, and a few pounds of matte containing most of the copper. The slag from the first run, averaging $27.48 per ton, is given a scavenger run, which reduces the value to $8.74 per ton. The base bullion is cupelled in an oil-fired furnace, producing a bullion 810 fine in gold and 140 fine in silver, impurities being largely copper and zinc with a small percentage of iron and lead.

One 600 cu. ft. per minute compressor, taking 240 H.P., supplies the 24 agitators with air at 28 to 30 lb. pressure. Power consumption is $1^1/_7$ H.P. per ton of ore milled per day. Mill solutions are carried at the equivalent of 1 lb. KCN per ton. Sodium cyanide, NaCN, is used. Cyanide consumption is .52 lb. per ton, of which .38 lb. is mechanical loss and .14 lb. chemical consumption per ton of ore milled. Lime is added to the ore at the No. 5 crushers from a chute from a 120-ton storage bin outside. Cyanide is dissolved and added to the boot of the concentrate elevator, keeping the solutions of this circuit at 1.5 lb. per ton. Additional lime is added to the concentrate circuit to raise the solution to 1.5 lb. per ton. Lime consumption is 2.45 lb. per ton of ore. The ratio of precipitation is 2 tons of solution to 1 ton of ore. Daily tonnage is obtained by taking the tonnage trammed plus or minus the decrease or increase in the bins, which are measured at 6 A.M. All samples are taken by the operators. One man on day shift cuts down, washes, dries, and prepares all samples for the assay office.

*Mill Organization.* — Mill organization consists of mill superintendent, assistant mill superintendent, reduction foreman, cyanidation foreman, repair foreman, 3 crusher shift bosses, 3 battery shift bosses, 3 concentration shift bosses, 3 solution shift bosses, and 3 repair shift bosses.

The assistant mill superintendent is responsible for daily operation and repairs. The reduction foreman has charge of all milling operations till the pulp overflows the classifiers. The succeeding treatment is supervised by the cyanidation

foreman. Crusher shift bosses have charge of the loading station at the 950-ft. level, of the crusher plant, and of the tram, and are responsible to the reduction foreman. Battery shift bosses have charge of conveyors, stamps, ball mill, tube mills, and classifiers, and are also responsible to the reduction foreman. Concentration shift bosses have charge of concentration, re-treatment of concentrates, agitation, and No. 2 mill thickening. Solution shift bosses have charge of sampling, classification, precipitation, filtration, and decantation, and assume responsibility in the absence of the foreman. Concentration and solution shift bosses are responsible to the cyanidation foreman.

*Extraction.*—Of the values extracted from the ore, 48.5% is dissolved in the tube mill circuit, 45% in the agitators, and 6.5% in the concentrate re-treatment circuit. An incidental extraction of between 2 and 3 cents per ton takes place in the decantation plant, chiefly in the last two tanks; this raises the value of the solution discharged in the pulp.

Undissolved values in the decantation tails for 1921 totalled $0.22 per ton, and dissolved values $0.08 per ton, giving a total value in decantation tails of $0.30 per ton.

The total mill tails were .28 cents per ton. With the installation of complete filter plant, the total tails will be reduced to .20 cents per ton.

*Costs.* —

| | | |
|---|---|---|
| Coarse crushing | $0.10 | per ton |
| Fine grinding: Stamps and tubes | .41 | " |
| " " Ball mill and tubes | .26 | " |
| Agitation | .03 | " |
| Concentration and re-treatment | .05 | " |
| Decantation | .03 | " |
| Filtration: Leaf filter | .12 | " |
| " Drum filter | .05 | " |
| " (Estimated for 3,000 ton plant) | | |
| Clarification, Precipitation, and Refining, per ton of solution precipitated | .04 | " |

The above costs include all labour with the exception of that of the mill foremen. Filtration costs include re-pulping of tails and pumping to tailing launder.

# MINING OPERATIONS AT THE McINTYRE PORCUPINE MINES, LIMITED.[1]

By The Staff.

Annual General Meeting, Ottawa, March, 1922.

The original McIntyre claims consist of two "forties" situated in lot 10 of Concessions I and II, township of Tisdale and on the Porquis-Timmins branch of the T. & N. O. Ry. The first production came from a small but spectacular vein near the south-west shore of Pearl lake. In March, 1912, a ten-stamp amalgamation and concentration mill was erected. In 1913 this was replaced by a 150-ton all-sliming cyanide plant. This plant has been gradually increased to a capacity of 525 to 550 tons per day, and additional units are now in the course of construction to bring the capacity to 800 tons per day. Meanwhile the original properties have been augmented by the acquisition of the Pearl Lake Gold Mines, Ltd., and the Jupiter Mines, Ltd., until the total area held in one block about Pearl lake is slightly over 360 acres. The early history of the mine is replete with financial difficulties, but a bold policy of expansion and far sighted recognition of the trend of economic areas has placed the company firmly on its feet.

## Geology.

The geology of the Porcupine area has been described by several writers, and is the subject of Part III of the Twenty-fourth Annual Report of the Ontario Bureau (now Department) of Mines, by A. G. Burrows, which contains the most exhaustive account available. The various rocks have been carefully described by A. R. Whitman in his paper, "The Rocks of the Porcupine District", printed in the *Transactions* of this Institute, vol. XVIII, 1915. No attempt will be made here to do more than barely outline the several formations found on the McIntyre properties.

(1) For an account of the general milling practice at the McIntyre mine, see paper by A. Dorfman, **Transactions**, Vol. XX, 1917, pp. 92-109. See also paper, by the same author, on "The Metallurgy of Carbonaceous Ores", immediately following this paper.

Figure 1. General view of the McIntyre Porcupine Mines.

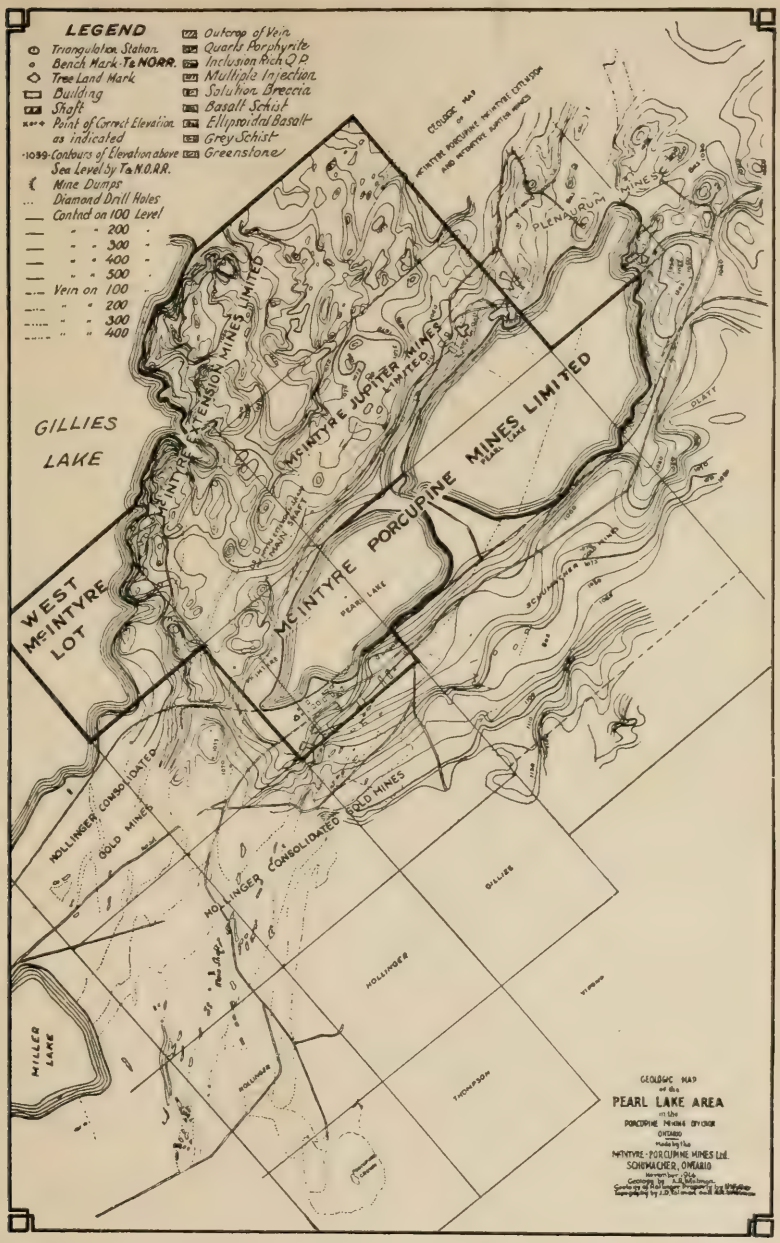



**Figure 2.** Geology of surface, from 1916 Annual Report.

The oldest formation is generally designated as the Basaltic-Greenstone series, and the types are variously described as basalt schists, pillow lavas, greenstones, sericitic and chloritic schists, and carbonate or grey schists; but all are apparently related rocks of similar basicity, averaging about 50% $SiO_2$. These rocks are completely altered mineralogically but they exhibit striking similarities to certain less altered basalt flows out-cropping to the south and east. Intrusive into the basaltic-greenstone series are masses of quartz porphyry of Algoman age. Several such occur in the vicinity, but a large lenticular mass, partly underlying Pearl lake, appears to be genetically connected with the McIntyre ore-bodies. In area it is approximately 125 acres, with a maximum east and west dimension of 6,000 ft. and a width north and south of 1,400 ft. It is schistose and considerably altered, but it is readily distinguished from the older formations by its lighter colour, prominent quartz phenocrysts, and higher silica content, ranging from 60 to 65% $SiO_2$. Owing to the close relationship between this intrusive and the ore-bodies, its peculiarities have been carefully observed. It has been found that the western end retreats eastward on each successive level, so that north and south vertical sections present a chonolithic form. The eastern end appears to have a parallel pitch, and other masses in the vicinity are known to show similar tendencies.

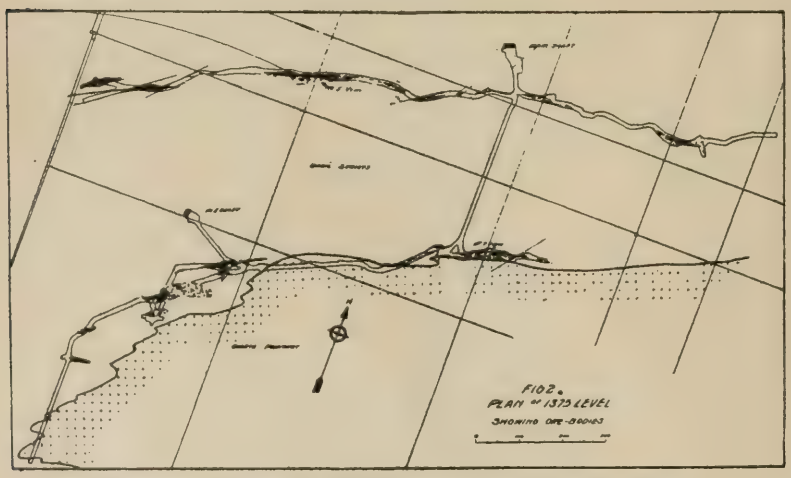



Figure 3. Plan of 1,375 ft. level, showing ore-bodies.

Veins are found in the porphyry, on the contact between the porphyry and the basic schists, and in the basic schists.

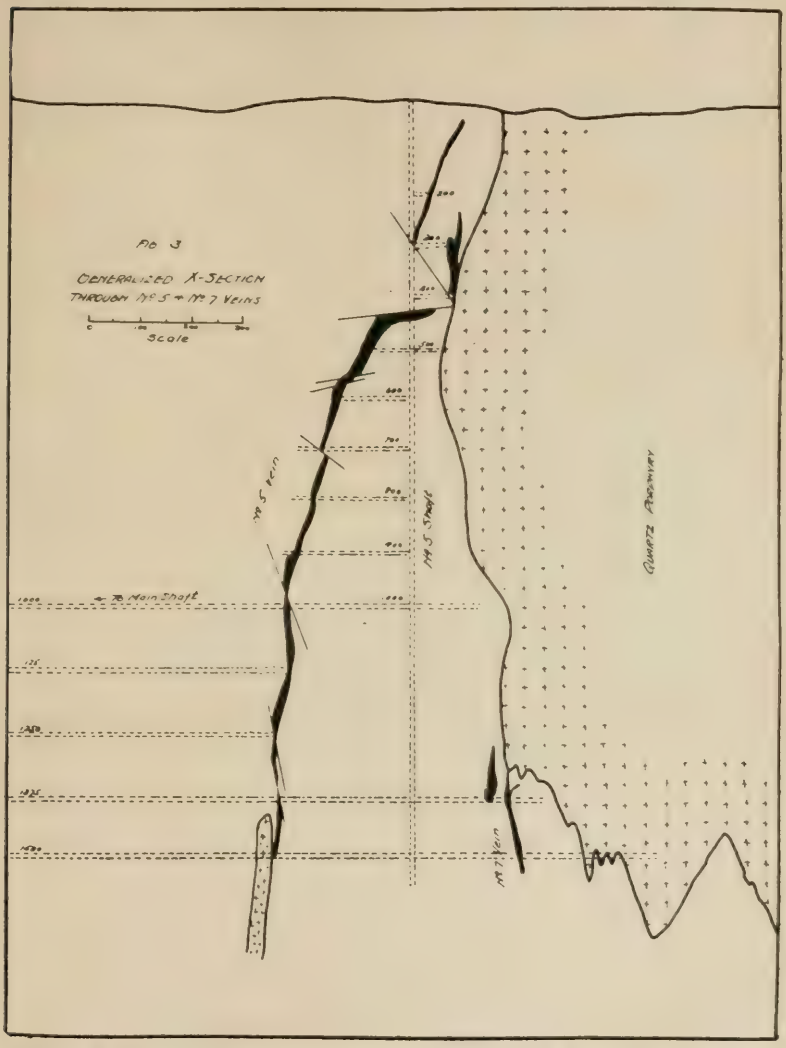



**Figure 4. Generalized cross-section through No. 5 and No. 7 veins.**

The same vein may traverse all three locations. Veins in the porphyry, however, are barren or too low grade to mine, with few exceptions. Veins may be classed according to their

structural or mineralogical character, and considerable refinement in classification is possible under both headings. There are two general types, fissure veins and lodes. The first type consists of one or more connected lenses located within a single fracture. The distinguishing feature is the continuity of the fracture, fault, or sheeted zone, in the absence of vein material. The outstanding example is No. 5 vein. This vein has pro-

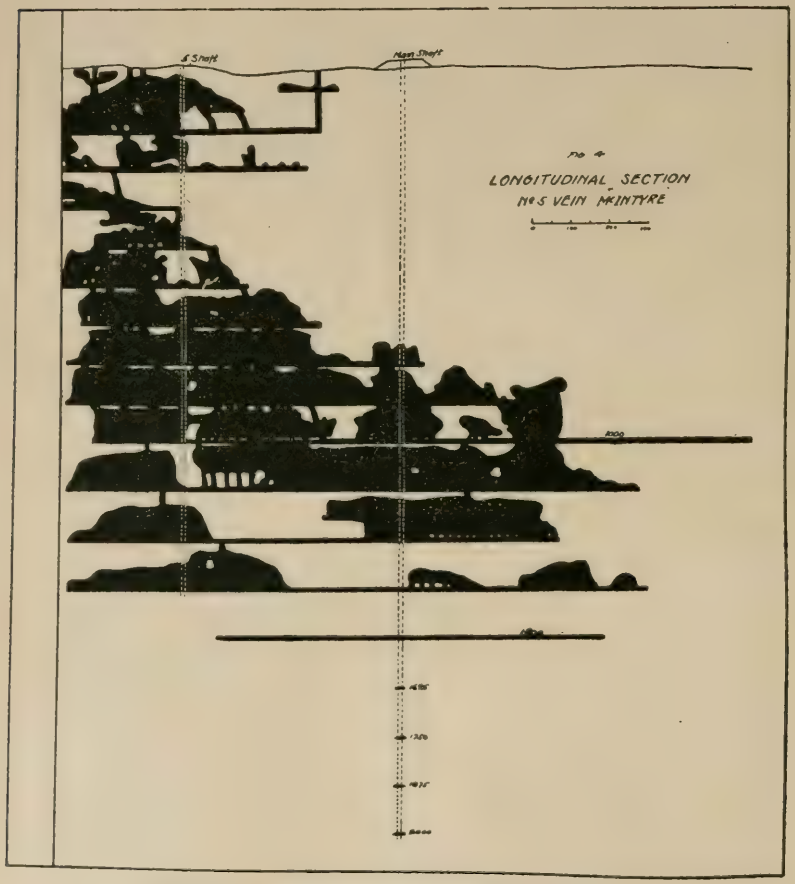



**Figure 5. Longitudinal section, No. 5 vein.**

duced over $8,500,000. It has been traced for a maximum length of 1,500 feet in the McIntyre mine, and has been mined and developed from the surface to the 1,500-ft. level. The

lode type is in the form, simply, of disconnected lenticular masses. They are arranged along general belts or zones and in echelon, the general rule being that each successive lens is offset to the left.

Two types of ore are readily recognizable, quartz ores and sulphide ores. Many veins of minor or negligible economic importance can be classified by the occurrence of such accessory minerals as microcline, chlorite, tourmaline, and sillimanite. The straight quartz ore is rather low grade, up to $5.00 per ton, with occasional coarse gold. It is increasingly rich as inclusions of pyritiferous schist are found, so that banded quartz and schist may assay well above the mill average. The sulphide ore is higher grade, and several of the rich stopes on No. 5 vein are largely of this variety. The best values are found in highly mineralized, dark, chloritic schist, carrying over 15% pyrite. Values from $50 to $100 are not uncommon for individual samples, and certain stopes consistently average over $15.00 per ton. Both types of ore may occur along the same vein. In general, the quartz ore occurs in the lighter coloured schists and near the quartz porphyry. Visible gold is rare, especially in the sulphide ore, high-grade specimens of which do not show any visible gold under high magnification. Sphalerite, galena, and scheelite are minor accessories, and are usually associated with coarse gold. Certain parts of No. 5 vein are intersected by "graphitic", or carbonaceous, faults, and the ore mined from these areas is contaminated with carbon. The metallurgical difficulties resulting from the presence of this material, and the treatment devised to overcome them, are described by Mr. André Dorfman in the succeeding paper.

The mill feed is a mixture of quartz and sulphide ores. The prominent gangue minerals are quartz, sericite, chlorite, Ca-Mg-Fe carbonates, and pyrite. A typical partial analysis is as follows:

| | |
|---|---|
| $SiO_2$ | 47.90% |
| Fe | 10.03 |
| S | 11.16 |
| $Al_2O_3$ | 5.14 |

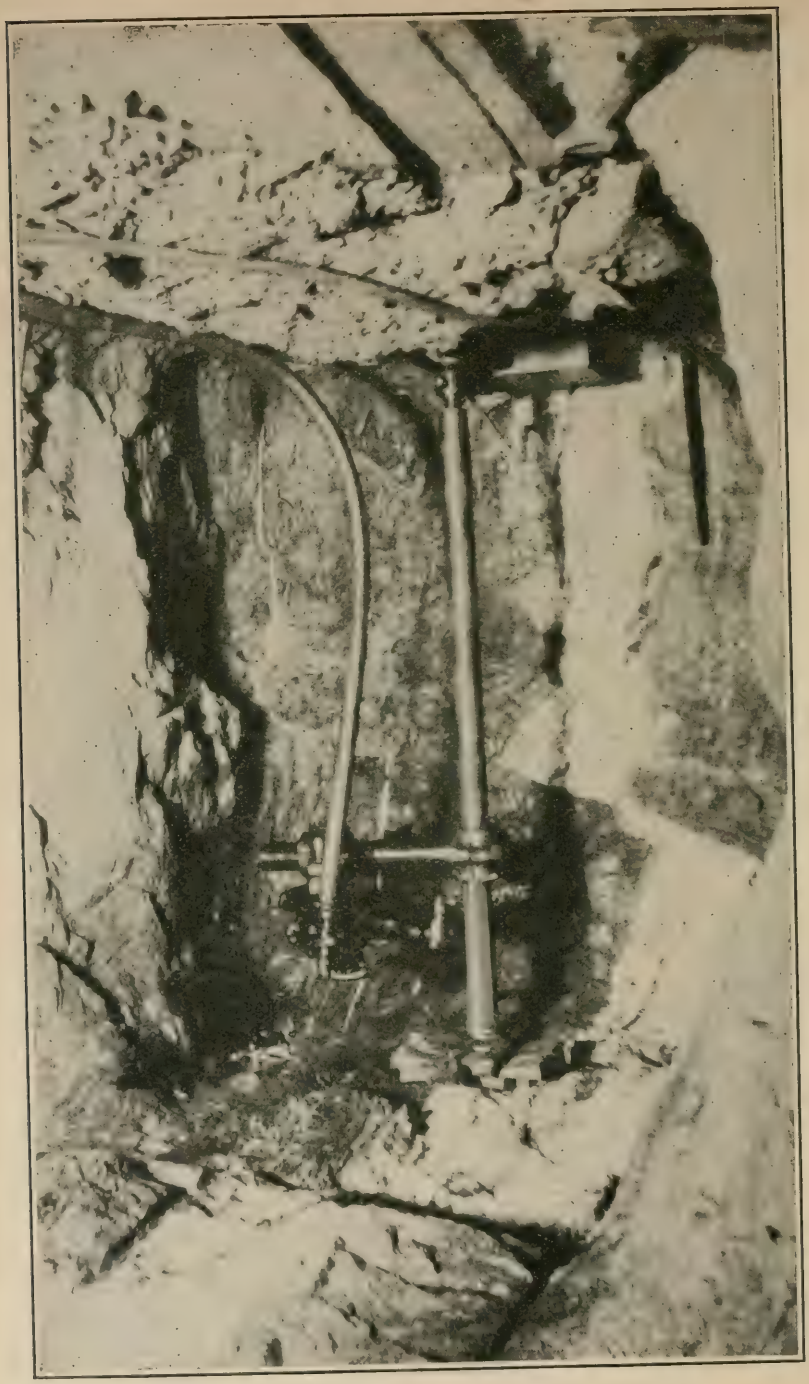


Figure 6. Faulted vein in 1523 drift, No. 7 vein.

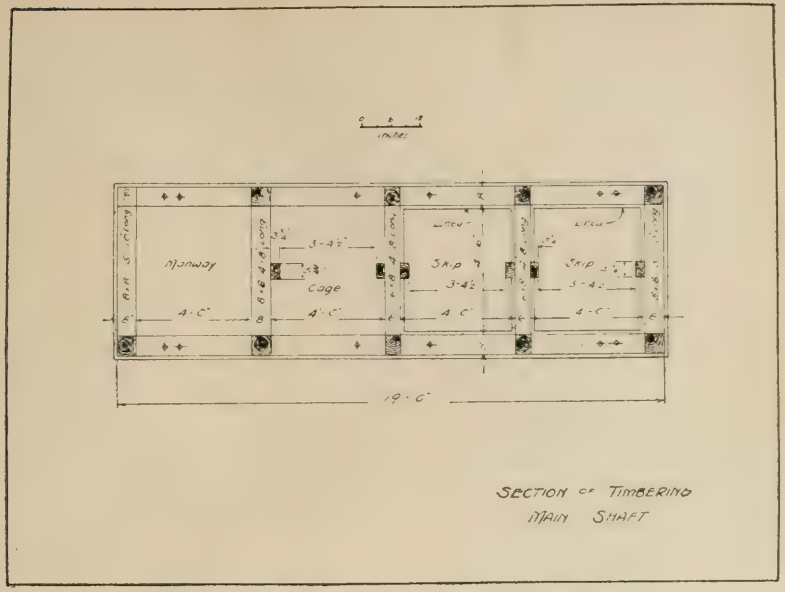



Figure 7. Section of timbering, Main shaft.

## MINING.

*Shafts.* — The principal shafts are:—

| Name | Compartments | Depth | Location |
|---|---|---|---|
| No. 4 | 2 | 800 ft. | South side of Pearl lake |
| No. 5 | 2 to 1,375 level<br>3 below 1,375 level | – 1,375 ft. | North side of Pearl lake |
| Main | 3 to 1,000 level<br>4 to 2,000 level | – 2,000 ft. | McIntyre Extension |
| No. 1 B | 2 | 1,000 ft. | Jupiter East claim |

At present only No. 5 and Main shafts are being operated. No. 5 shaft is used for men and material and Main shaft to hoist ore and waste. Main shaft has just been sunk to the 2,000 ft. level, and the work of cutting the ore pocket below the 1,875 ft. level is being completed. No. 5 shaft was sunk from the 1,000 ft., and raised from the 1,375 ft., level, and sinking is now being continued from this latter level. In the Main shaft the sinking compartment is located in the west end. The first operation is to sink this compartment 20 ft. and to drift under the cage and skip compartments, leaving a rock pentice of 12 or 13 ft. The shaft is then sunk full size

20 or 30 ft., and bearing timbers located below the pentice. At this stage steps are taken to collect all of the water coming

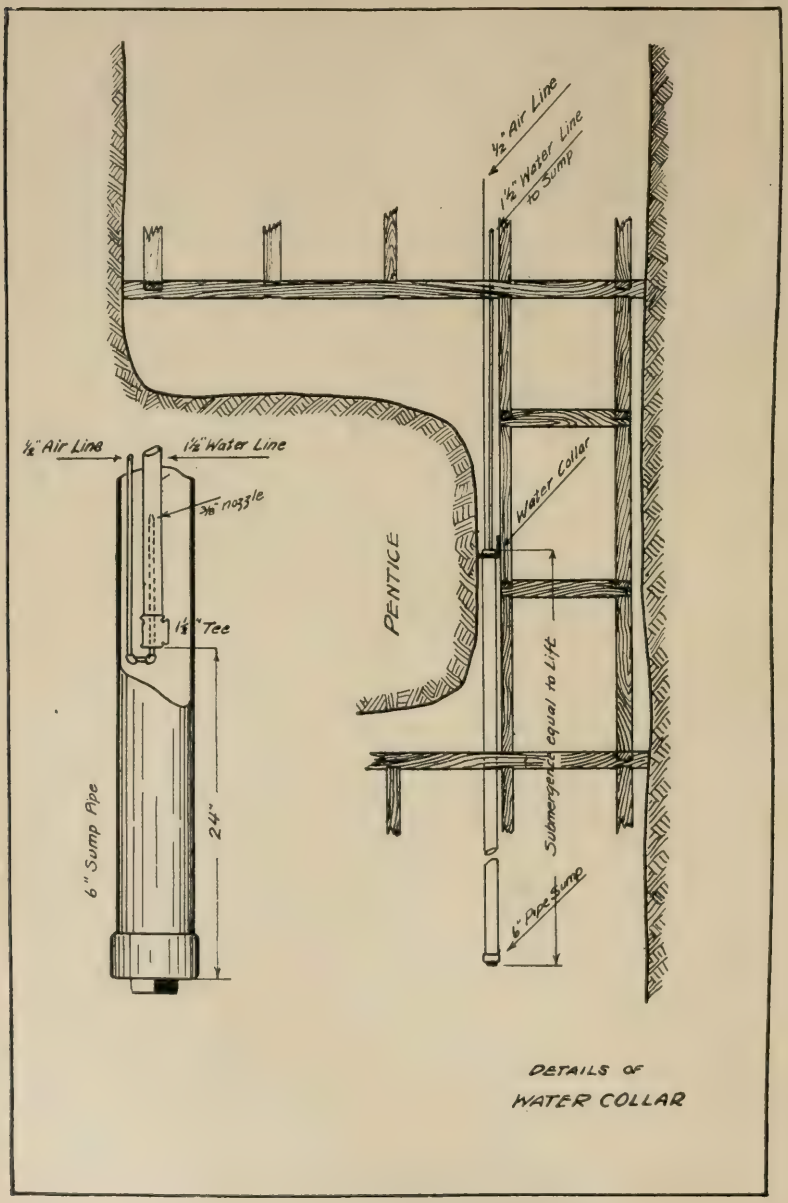



Figure 8. Details of water collar.

down the shaft. The partition between the sinking and the adjacent cage compartment is planked tight, and the water diverted from the sinking compartment above the sheave wheel. A dam is built around the sinking compartment at about the elevation of the middle of the pentice. The water so entrained is removed by a small air-lift. As no water has been encountered in sinking below the 1,000 ft. level, the shaft is kept perfectly dry by this means, which adds greatly to the efficiency of sinking operations.

The regular shaft crew consists of 5 miners (one acting as shift boss), 1 deckman, and 1 hoistman, for each 8 hour shift. Waugh Model 60 drills are used, dismounted and provided with handles and shock-absorbing springs. The shaft sets are placed by the regular shaft crew, and are carried well above the bottom, unless the ground is inclined to scale. The last set is protected by a blasting set, armoured with ⅜ in. sheet steel on the under side and slung under the last set with two one-ton chain blocks. The blasting set forms a working platform when new sets are being placed. Bearing timbers are placed below each level, and midway between each level. Level stations are cut as passed, as is the loading pocket below the main haulage level; but the ore passes, waste pass, ore pocket, etc., are left until sinking is completed, so that the large tonnages of waste can be hoisted by skips. Progress varies from 20 to 25 ft. per week, the work being divided as follows:—

| | |
|---|---|
| Drilling | 20% of total time |
| Mucking | 45% " |
| Timbering | 20% " |
| Blasting and miscellaneous | 10% " |

When sinking is completed, the guides are removed from the sinking compartment and placed on the cage and skip compartments, and the manway, previously located in one of the skip compartments, is changed to the sinking compartment. The pentice is then drilled off with stopers, using steel long enough to reach to within 2 to 2½ ft. from the top. Holes are given a very light burden, 14 to 18 inches, and shot a few at a time, the muck falling to a platform built over the set imme-

diately below the pentice. The muck is removed, and the timber and guides connected through. Skips and cage must be hung up while the pentice is being removed.

Sinking operations are similar in No. 5 shaft. A maximum of 35 ft. per week was attained during the sinking of the two compartments below the 1,000 ft. level. During raising, the shaft was carried up full size. A cribbed manway occupied one end, and the rest of the shaft served as a pocket, with a loading chute at the 1,375 ft. level. Two plugs were located in the walls of the 1,375 station, and a string stretched between

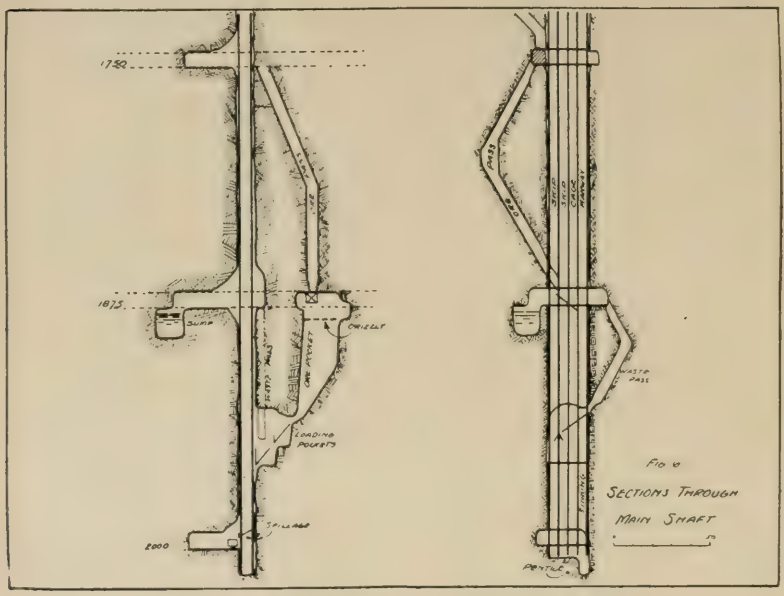



**Figure 9. Sections through Main shaft.**

them established the center-line of the shaft and could be transferred up the raise at any time by plumb lines. Rand CCw21 wet stopers were used. A diamond drill hole was put down from the 1,000 ft. level to promote ventilation, but failed to break through owing to deflection. This work has demonstrated that sinking operations are faster, more economical, and generally more satisfactory, than raising. The shaft and the raise met perfectly, the error, as established by closure on a single wire, being North 0.02 ft., East 0.12 ft.

*Spacing of Levels.* — Above 1,000 ft., levels were driven from No. 5 shaft at 100 ft. intervals. The spacing below the 1,000 ft. level has been increased to 125 ft., which distance has

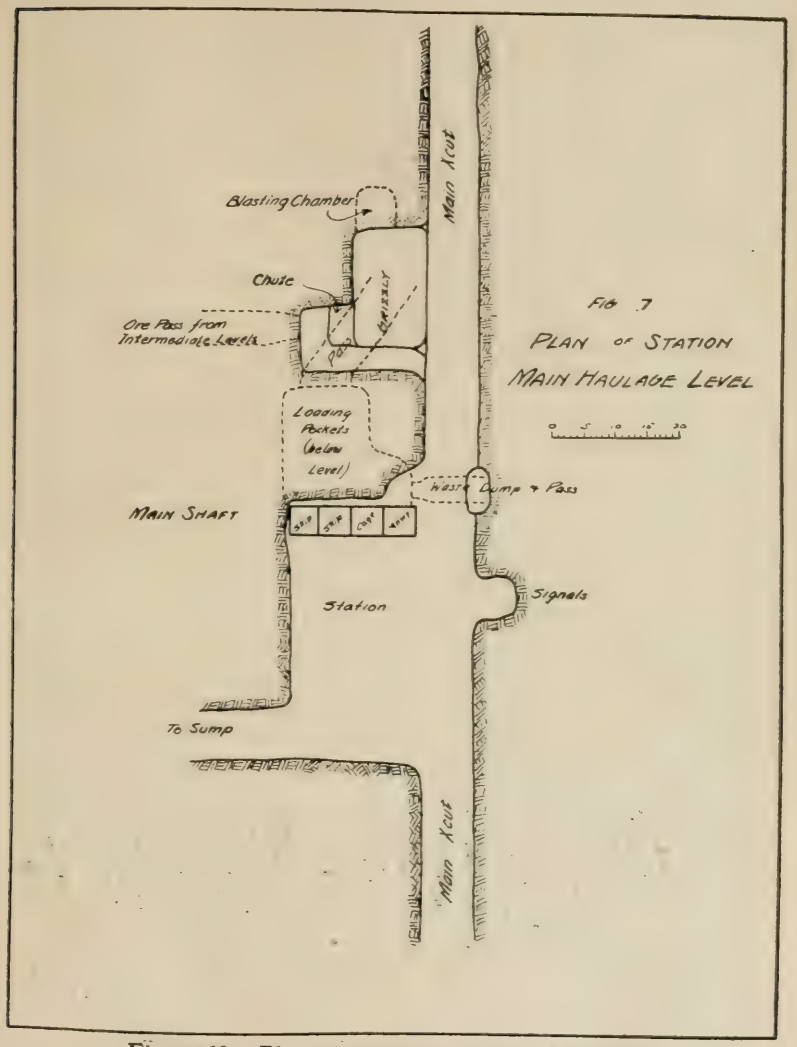



**Figure 10. Plan of station, main haulage level.**

been found to be a convenient compromise between the desire to minimize development and the restrictions placed by irregularities of the ore-bodies and the exigencies of mining. The

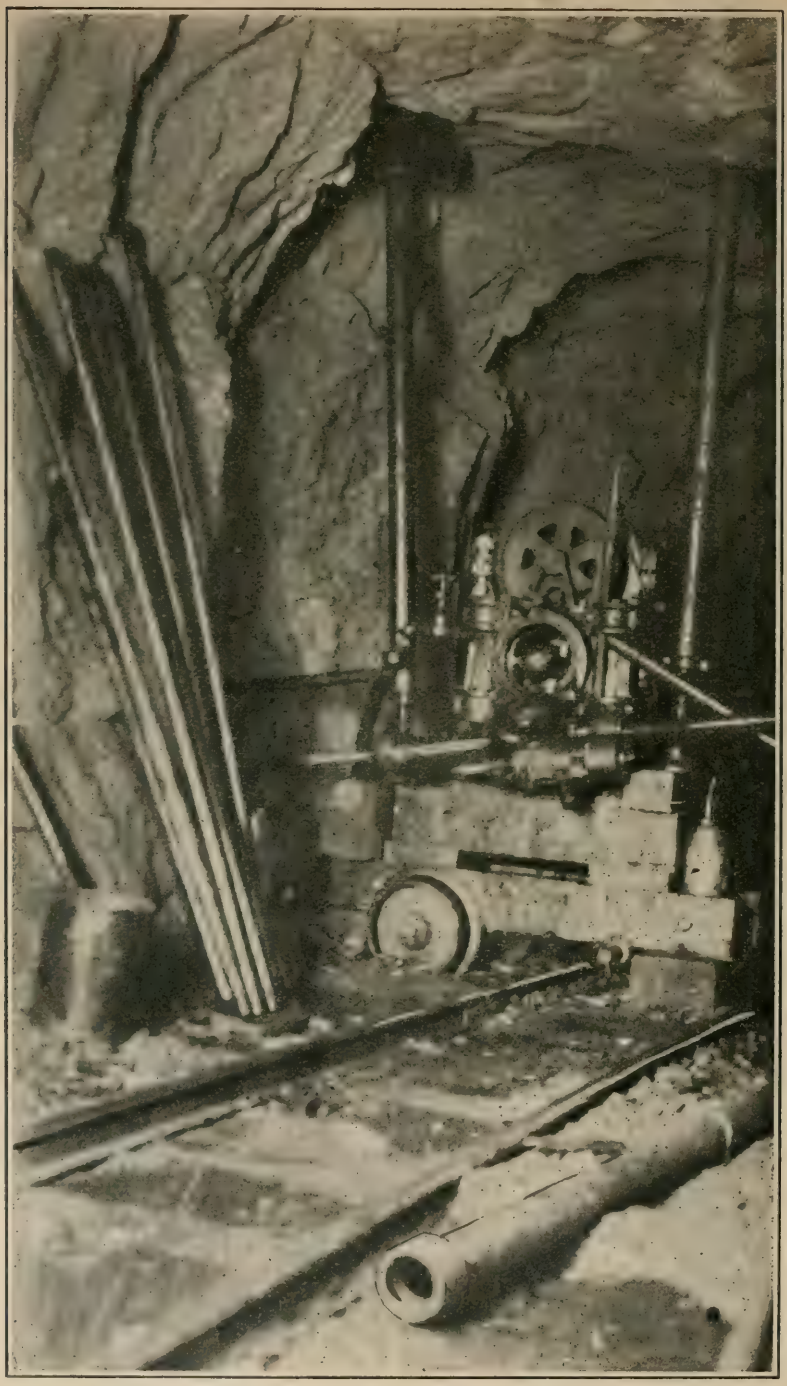


Figure 11. Beatty Bullock diamond drill. Set-up in drift for short flat hole.

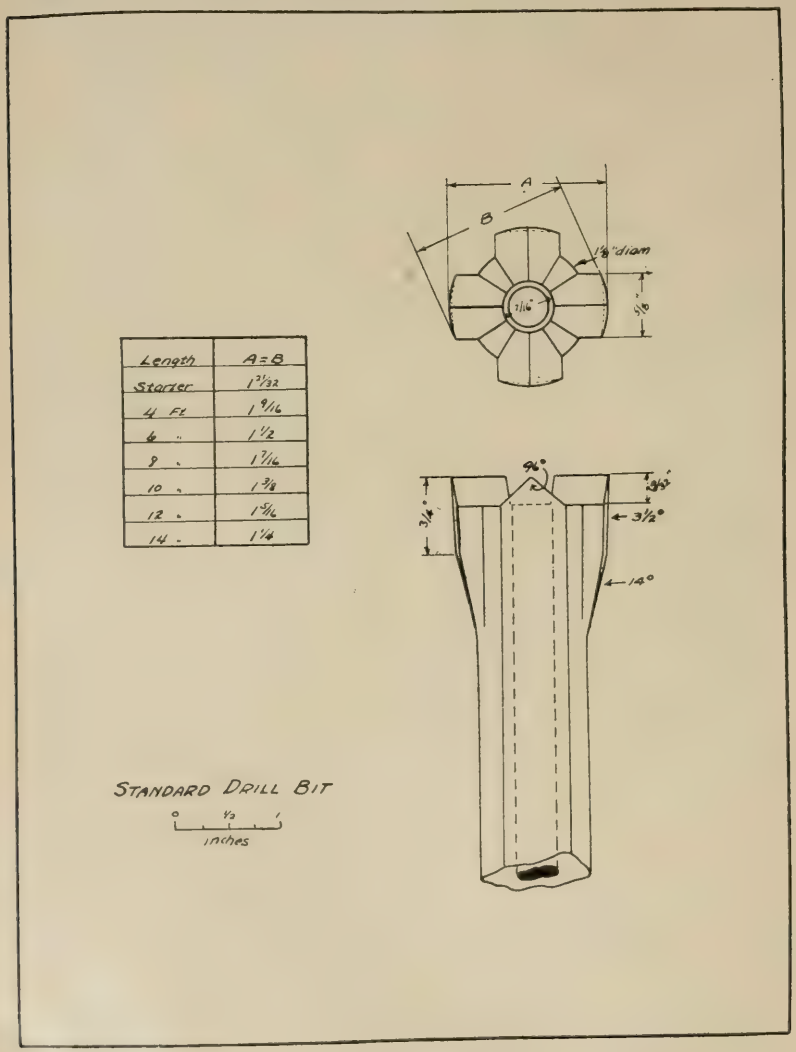
1,000, 1,375, and 1,857 ft. levels are designed as main haulage levels, and are equipped with ore pockets at the Main shaft of approximately 400 tons capacity. The 1,000 and 1,875 ft. levels also have waste pockets, but of smaller size. All ore is dumped over a 14-inch grizzly above the ore pockets. The 1,125, 1,250, 1,500, 1,625, and 1,750 ft. levels are intermediate levels. The first two have been connected with the 1,375 by ore passes which convey their production to the 1,375 ore pocket. Similarly, the 1,000 ft. ore pocket is served by the levels above, and the 1,875 ft. ore pocket will be served by the 1,500, 1,625, and 1,750 ft. levels. Ore passes are raised from level to level on a 55 degree slope with one bend, the bench of which is enlarged to prevent blocking. By providing each raise with a chute they act as subsidiary pockets.

**Figure 12. Set-up in 1521 drift, No. 7 vein.**

*Development.* — Development work is guided by rather extensive use of diamond drilling. Long inclined holes are drilled below the lowest working levels in order to gain information on which to plan future development. For this work a Longyear No. 2 machine of 2,000 ft. capacity is used. A great many short holes are drilled to locate parallel lenses, faulted ore-bodies, and contacts, and to direct stoping. Either

a Sullivan S or a Beatty Bullock machine is used with an E bit. About 600 ft. per month is the average.

All drifting and cross-cutting is let out on contract. Drifting commands the higher price, the other variables being the length of tram and the conditions of waste disposal. It has




| Length | A=B |
|---|---|
| Starter | 1 3/32 |
| 4 Ft | 1 9/16 |
| 6 " | 1 1/2 |
| 8 " | 1 7/16 |
| 10 " | 1 3/8 |
| 12 " | 1 5/16 |
| 14 " | 1 1/4 |

Figure 13. Standard drill bit.

become standard practice to drill off, with one machine, on a single shift, and to blast on that shift. The face is mucked out on the opposite shift, and the muckers set up the drill and sometimes drill off one or two holes. A pyramid cut is used with 20 to 24 holes to break an 8 by 8 ft. drift. The average round is just under 6 ft. The cut is always blasted alone and inspected before loading and blasting the square-up. The standard machine is the Waugh Model 60; several Rand 88's are in use. One-inch hollow hexagon steel, which has been the standard, is gradually being replaced by one-inch quarter octagon. The modified octagon section is less liable to stick in the drill chuck and is a generally better section. A bit of the type shown in Fig. 13 has been adopted lately. Starters measure $1\frac{23}{32}$ inches, and the gauge reduction is $\frac{1}{16}$ inch per foot. A repair shop is located on the 1,000 ft. level near the Main shaft. Drills may be tested there under working conditions before being returned to the drill-runner. Waugh Model 60's and 88 Rands must drill 13 inches per minute on an air consumption of not more than 125 cu. ft. per minute at 100 lb. Any drill that fails to stand up to this test is again examined for worn parts.

*Stoping.* — The ore-bodies are well adapted for shrinkage stoping. The veins are steeply dipping to vertical, but they are too irregular and too narrow to be mined by any modified caving system, and are ill-suited to immediate complete extraction. On the other hand, the walls are fairly good, and aside from scaling slightly on exposure, will stand with little support beyond a few stulls where the ground is weakened by joints or faults. Stopes have remained open on the upper levels for years with only minor sluffing of the walls. There is some evidence of increased pressure on the lower levels, however, and it is recognized that waste filling of mined out areas may have to be resorted to.

Two systems of opening up stopes are in use, employing timbered and solid backs. The determination of the particular method to be employed depends on the width, grade, and location of the ore-body. As drifts advance along the veins, test-holes are drilled into the walls and the cuttings assayed. If the

test-holes are bottomed in waste, a narrow ore-body is delineated; if in ore, widths greater than 18 ft. may be expected. In case of narrow ore-bodies, the first step is to slash out to the full width of the vein. Next a cutting-out stope is mined to a height of 16 to 18 ft., and mucked out. This work is usually contracted. Timbering follows about 100 ft. behind the stoping. Stulls are placed 5 ft. centre to centre, and chutes built on 25 ft. centres. The standard chute is shown in Fig. 14.

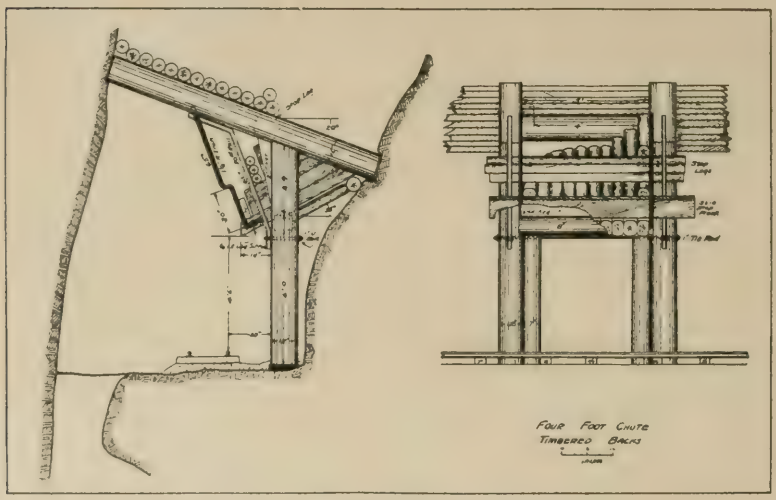



**Figure 14. Standard chute, timbered backs.**

Stulls are hitched deeply into the foot-wall and placed against the hanging with a strong batter, in order to be self-sustaining when the floor pillar is removed. Stulls are further supported with one or two posts in order to resist deformation during the stoping period. If the ground is too weak to furnish a safe hitch, a horizontal cap supported with two posts and wedged tightly to the walls is used. The first round above the timbers is drilled with steeply inclined holes so that the back is blasted against the muck and not slammed down on the unprotected lagging.

The procedure in the case of large ore-bodies depends largely on the grade of the ore, but is influenced also by its location

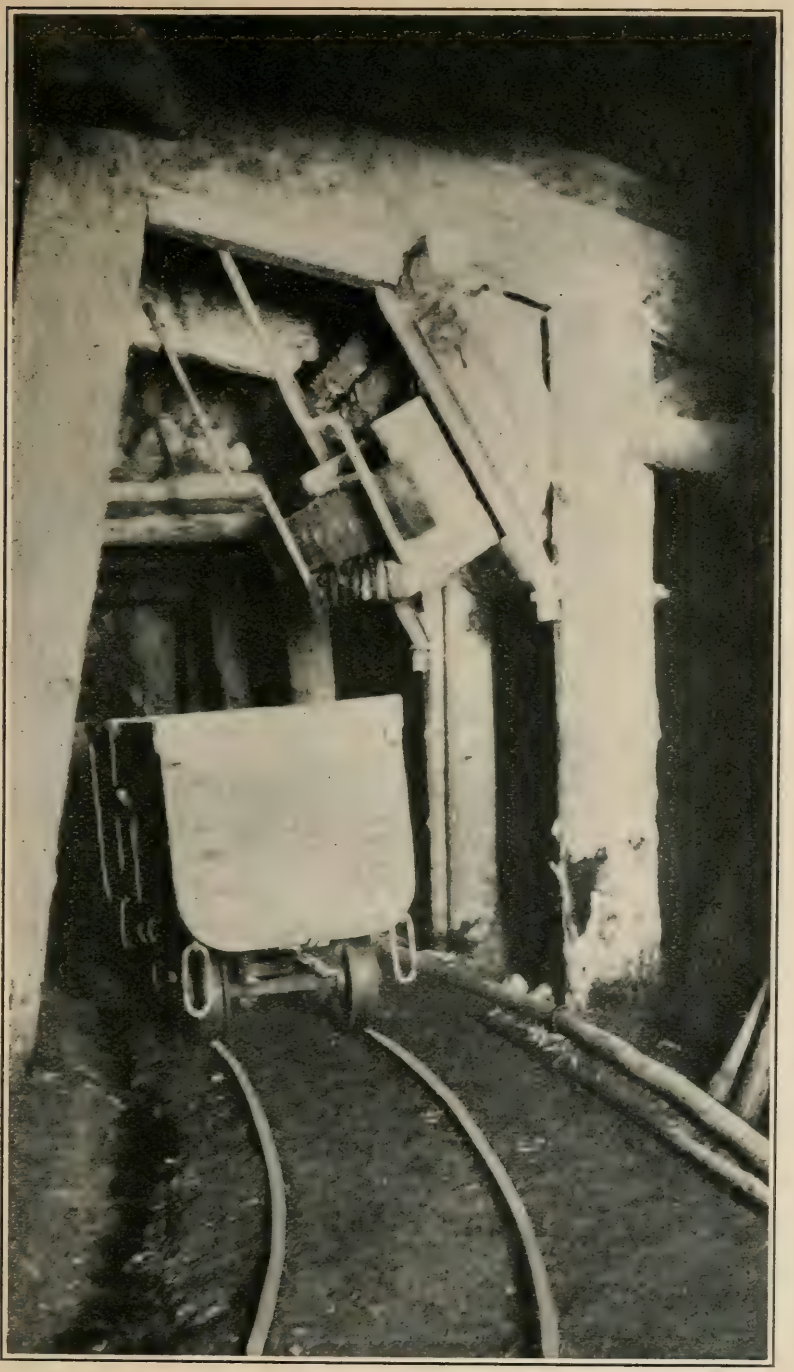


Figure 15. Standard chute, timbered back.

and by the nature of the ground. If the grade is high, the ore may be slashed out the full width of the lens and a cutting-out stope made, as in the case of a narrow ore-body. Faulted or jointed ground may demand the same treatment. The advantage is the quick realization of ore otherwise tied up in pillars, and the disadvantage is the difficulty of removing the floor

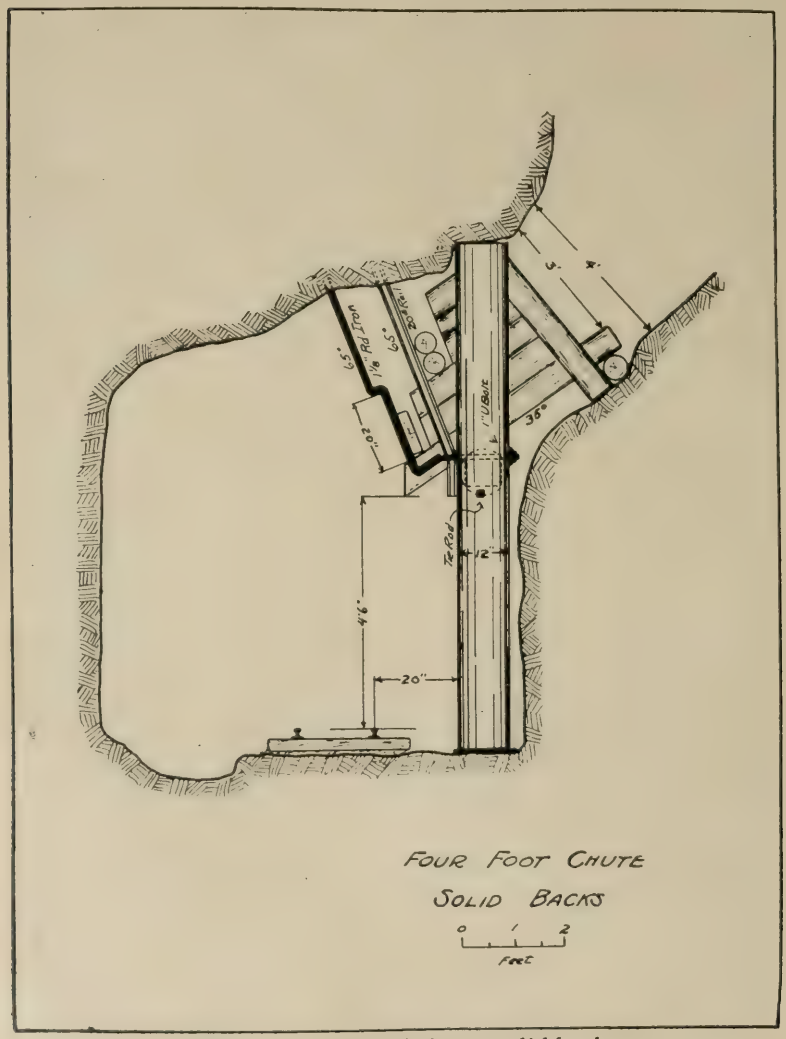



Figure 16. Standard chute, solid back.

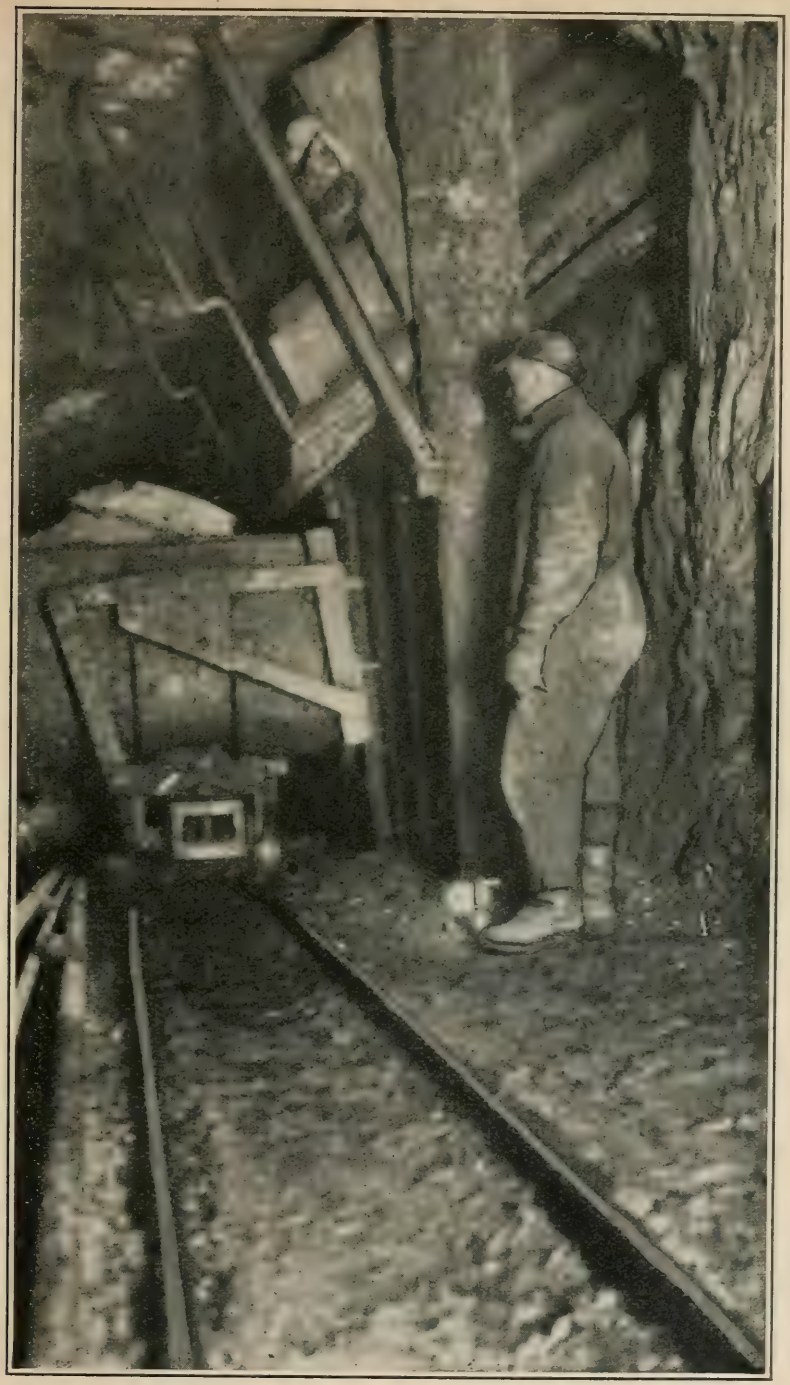


Figure 17. Standard chute, solid back.

pillar below a mass of timbers which are not self-sustaining. Usually, wide lenses are opened up by a series of box-hole raises, spaced 25 ft. centre to centre. The raises are 24 ft. high and are connected at the tops by narrow sub-level drifts, leaving a solid back of ore about 10 ft. thick. Standard chutes are built into the box-holes. The tops of the box-holes and the sub-level drifts are then enlarged to the full width of the ore-body.

Whatever particular method is used to start the stope, the actual operation of stoping does not vary greatly. The wider stopes more often require pillars, though this condition is generally the result of the sudden narrowing of the ore-body or flattening of the dip. Manways are provided at intervals of

Figure 18. Benching, 1226 stope, No. 5 vein.

200 ft., though shorter stopes are equipped with two manways. If the stope is inclined, the cross-section of the manway is small, 4 by 4 ft., and the ladders are continuous; but if the stope is vertical, it has been found advisable to install 5 by 5 ft. (inside) manways with landings. All manways are built of half-round spruce cribbing, framed in the saw-mill. By the time the

stope has been mined up 60 to 80 ft., a raise is driven through to the next level, to promote ventilation and facilitate the handling of material. The standard machine is the Waugh Model 60, mounted on column and arm. A typical bench (say 9 ft. wide) is carried 8 ft. high. Three rows of three 9-ft. horizontal holes are drilled from a single set-up. The vertical burden is 2½ ft. and the horizontal spacing is about 4½ ft. In wider stopes the machine is set up on the footwall, and holes are fanned out towards the hanging. They are loaded with, as an average, 8 sticks of 35% powder per hole. The average duty for last year, including all preliminary work, was 35.6 tons per machine shift, with a marked increase in efficiency during the last six months. The normal duty, underground working conditions, is now about 85 ft. per machine shift. Powder used in stoping operations, including breaking down, block-holing, and chute blasting, amounts to 1 lb. per ton. Very little block-holing is necessary, and Rand Jackhammers are used for such work. A floor pillar varying in thickness from 6 to 15 ft., according to the width of the stope and the nature of the ground, is left between the back and the level above. It has been found absolutely necessary, for safe working conditions, to remove the pillar with one row of holes.

If the stope is narrow and not strictly vertical, the walls must be mucked down as the final ore is drawn, as it hangs up on the footwall tenaciously. The back is first carefully scaled, and in some cases lagged over. The footwall is mucked clean as the ore is drawn, and any loose ground is scaled or caught up with light sprags and stulls. Stopes are not completely emptied, as some ore is left to protect the lagging when the floor pillar above is blasted down. After this is done, the chutes are drawn clean and the ore between the chutes is recovered by building a sollar, under which a car can just pass, and cutting a hole in the lagging. Work retreats away from the far end of the stope. The line of stulls and lagging left over each drift soon become loaded with waste, which sluffs off the walls of the stope and forms a good barrier against heavier rock falls. With solid backs, once the floor and roof pillars are removed, the opening is continuous, unless pillars of ore are definitely abandoned in order to support the walls.

Occasionally, batteries of timbers are placed where the value of the ore left would be excessive, or where the pillar in ore would be weak. At other times it has been found necessary to lag the under sides of the pillar. Throughout stoping, and especially during the removal of floor and roof pillars, there is some dilution of the ore with waste. The average dilution as revealed by discrepancies in tonnages broken and extracted from carefully measured stopes, is about 10%. Stoping is controlled by mineralization and assays, and the indefinite boundaries of the ore-bodies tend to dilution of values.

Figure 19. Loco, Granby car, and tipple, 1375 ore pocket.

*Tramming* — Side-dump Hudson cars of 20 cu. ft. capacity are used on the intermediate levels, where hand tramming is the general practice. On the main haulage levels, 38 cu. ft. automatic side-dump Granby type cars are used and are hauled by storage locos. Two locos are in use, a 1½-ton Automatic Transportation Company unit, and a 2½-ton Whitcomb, the latter equipped with 40 type A10 Edison storage batteries of 375 ampere capacity. Charging is done on the blasting shifts. The charging equipment is located on the main haulage level, and consists of a 17KW, 125 volt, 136 ampere

Canadian Westinghouse DC generator, AC motor, and the usual switchboard equipment. The normal charge is 75 amperes. Twenty-pound rails are used throughout the mine, laid to 0.5% grade, but it is planned to use 30 lb. rails in the future on the main haulage levels. Curves are laid to a minimum radius of 30 ft. where motor haulage prevails.

*Surveying and Sampling* — All contracts are measured up weekly. Development work is surveyed to date at least once a month. Stopes are measured once a month, and are surveyed before being drawn empty and at such other times as seems advisable to direct mining. Sampling is an important part of the daily operations, and the closest co-operation between the engineering and operating departments is maintained in this work. Development faces are sampled after each round. Backs of stopes are sampled at 10-ft. intervals along each bench. Grab samples (box samples) are taken from all ore trammed out. Development samples reach the assay office before noon each day, and assays are available by five o'clock. They are on file with the mine superintendent, together with stope samples and box samples of the previous day, by the next morning. Box and channel samples check fairly well, but both are invariably higher than the mill returns. In channel samples this is due to dilution of the ore and to the friable nature of the higher grade sulphides; in box samples it is probably due to the tendency to get too high a percentage of fines in the sample. The average sampling error is about 15%.

*Underground Organization* — Except for shaft sinking, all work is conducted on two shifts. Drilling is about equally divided between the two shifts. Timbering and track-laying is all-day shift. Stope blasting is done between shifts by a blasting crew. Drilling in the stopes on company time is inspected and measured at the end of each shift by a drilling foreman. This innovation has had a pronounced effect on the drilling efficiency.

*Safety Work* — Safety work is in charge of a safety engineer, who devotes the whole of his time to this work. The

company employs about 300 men, 200 underground and the balance on the surface. Since 80% of the accidents occur underground, the greatest amount of time and effort to prevent them is spent there. The safety engineer examines all the working places in the mine daily, and makes out a written report of conditions, with necessary recommendations, in triplicate. One report goes to the general manager, the second to the superintendent of the department interested, and the third to file. The superintendent's copy is posted in the shift bosses' office and must be initialed by each shift boss every day. While this work is of considerable importance, it is recognized that the best way to reduce the accident record is to inculcate in each workman the vital necessity of making their working conditions safe at all times. The fact that a qualified man examines these conditions daily, and is open at all times to suggestions for their betterment, makes for co-operation in safety between the company and the individual workman. In conjunction with this work goes the usual Safety First propaganda, carried on by means of Safety First literature and bulletins. There has been a steady decrease in the number of accidents per year for the three years that the safety work has been carried on.

*Pumping* — About 30,000 gallons of water are pumped from the mine per day. The principal sources of water are surface seepage, and light flows from broken, sheared, and faulted ground on the 600, 1,000 and, to a minor extent, the 1,375 ft. levels. Pumping stations are located and equipped as follows:—

| Level | |
|---|---|
| 600 ft. | one 5½ by 10 electric-driven vertical triplex Aldrich |
| 1000 ft. | one 5½ by 10 " " " " |
| 1500 ft. | one 3½ by 5 " " " " |
| *1875 ft. | one 5 by 10 " " " " |
| *2000 ft. | one 6 by 4 by 6 air-driven duplex |

* To be installed.

*Hoisting.* — A 9 by 12 Jenckes Machinery Co. Hoist is used for sinking the Main shaft. Muck is hoisted in 15-cu. ft. buckets equipped with the Sargeson safety cross-head.

A 9 by 11 Lake Shore Engine Works hoist is used to sink No. 5 shaft. Two Kimberley skips, of 49 cu. ft. capacity, operate in balance in the Main shaft. The hoist was designed by Nordberg. It is geared to a 350H.P. Canadian Westinghouse induction motor. Specifications call for the hoisting of a maximum net load of 6,000 lb., two 4,500 lb. skips in balance,

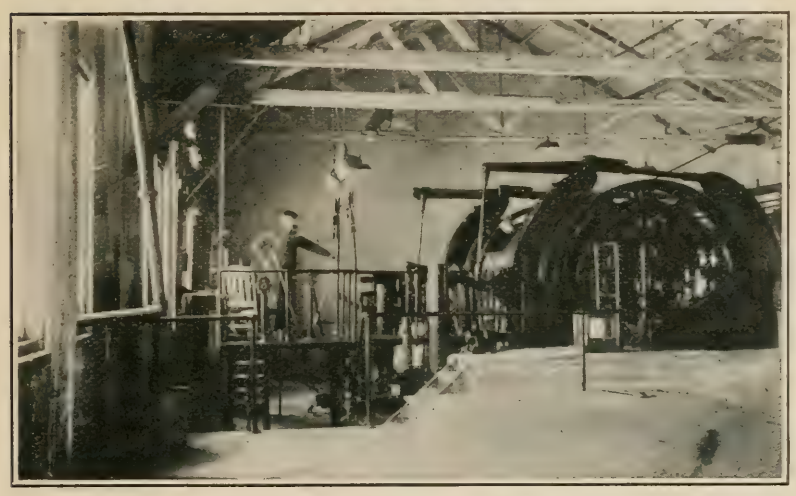


**Figure 20. Nordberg hoist, Main shaft.**

with a rope speed of 1,500 feet per minute, from a maximum depth of 2,700 ft. Drum diameter is 7 ft.; 1⅛ inch cable is used. No. 5 shaft is equipped with a Nordberg hoist, geared to a 125H.P. Crocker and Wheeler induction motor. A 3,000-lb. cage is hoisted in balance with a 3,750 lb. reticulated counterweight. Diameter cage drum, 6 ft.; diameter counterweight drum, 5 ft.; cable, 1 inch; rope speed, 750 ft. per minute.

A modern system of electrical signalling, together with a telephone system, is installed in each shaft. Cagers can be called with flash or buzzer signals, which do not repeat in the hoist-room; but only the cagers and the skip-tenders are permitted to signal to the hoistman, who must repeat all signals before moving the hoist.

*Compressor Plant.* — Three electrically driven compressors are in use, with a steam driven Ingersoll-Rand unit in reserve. The principal machine is an Ingersoll-Rand horizontal cross-compound compressor, rated at 3,000 cu. ft. free air per minute at 100 lb. It is driven with a 480 H.P. synchronous motor. This machine is supplemented with two Alley and MacLellan vertical compound compressors, rated at 600 cu. ft. per minute each and driven by a 125H.P. General Electric induction motor.

*Shops.* — The steel sharpening shop is located conveniently near No. 5 shaft. It is equipped with two Leyner sharpeners and the usual power hammers and forges. The heating and tempering forge is coke burning. The average duty is about 600 pieces of steel per day. Mine and shaft timbers are cut and framed in the sawmill, located between Main and No. 5 shafts.

# ON THE METALLURGY OF CARBONACEOUS ORES.

By André Dorfman

Annual General Meeting, Ottawa, March, 1922

During the month of September, 1914, the mill tails at the Porcupine Crown mines, Timmins, Ontario, apparently without any reason, showed considerable increase. This was the more surprising, seeing that the extraction at this plant, where a combination of amalgamation and cyanidation was practised, had never before dropped below 95%. The cause of the loss in extraction was soon determined by the operators, and traced to the presence of carbonaceous rock in the mill heads. A sample of the material (locally called graphitic schist) was submitted to the writer for examination.

Upon bringing the pulverized sample in contact with a gold cyanide solution, it was found that the material acted as a powerful precipitant. Precipitation tests were then carried out, by percolation and agitation, on crude and roasted samples, all reduced to pass a 100-mesh screen. Percolation was performed in a 6-in. Gooch funnel. Forty grams of the sample were spread over a double filter paper, and 100 c.c. of solution poured over the pulp; when this solution had percolated through, another 100 c.c. of the same grade solution was thrown on the filter. This operation was repeated five times and the tails from each percolation were collected separately. Percolation took place in 5 minutes.

Five similar tests were made on roasted rock. The agitation tests were made in bottles, using only 2 grams of the crude or roasted sample.

The original solution value used in each test was $2.64 per ton. The assay results are as follows:

**Percolation Test on Crude Sample.**

| | | |
|---|---|---|
| 1st 100 c.c. | (percolated) | $0.012 per ton |
| 2nd 100 c.c. | " | 0.012 " |
| 3rd 100 c.c. | " | 0.012 " |
| 4th 100 c.c. | " | 0.012 " |
| 5th 100 c.c. | " | 0.036 " |

**Percolation Test on Roasted Sample.**

| | | |
|---|---|---|
| 1st | 100 c.c. (percolated) | $2.64 per ton |
| 2nd | 100 c.c. " | 2.64 " |
| 3rd | 100 c.c. " | 2.64 " |
| 4th | 100 c.c. " | 2.64 " |
| 5th | 100 c.c. " | 2.64 " |

*Agitation Tests.* — The sample, when agitated with a pregnant gold solution, acted as a precipitant almost instantaneously, and on further agitation for 48 or 72 hours no values were redissolved. The roasted sample, however, lost all its precipitating power,

The above assays very convincingly show the harm resulting from the presence of carbonaceous rock in an ore treated by the cyanide process.

While the Porcupine Crown was the first mine to encounter "graphitic" schist, the other mines of the camp soon met the same obstacle. Thus, at the McIntyre, the ore-bodies were found to be increasingly contaminated with carbonaceous matter with the development of the lower mine levels, and on each level blocks of ore, although carrying excellent values, were "blackened" by graphitic schist, and had to be excluded as a source of supplying mill feed until a satisfactory method of treatment was found, and the plant changed and adjusted for such treatment.

## DETAILS OF EXPERIMENTAL WORK

The amount of gold precipitated by carbonaceous schist from a sodium auro-cyanide solution ($NaAu(CN)_2$), appears to be in direct proportion to the quantity of carbon present in the schist. Whatever are the chemical or electro-physical reasons for this reaction, the amount of gold precipitated corresponds quantitatively to the carbon content. The percentage of carbon in the carbonaceous matter not being constant, the actual quantity of gold precipitated was determined as follows:

Five grams of the finely pulverized graphitic schist, containing 5% of carbon, were agitated for 24 hours with 300 grams of pregnant solution, assaying $6.14 per ton, after which period

the pulp was filtered and the solution assayed. As the assay of this solution showed $1.56 per ton, the precipitating power of the schist per ton of ore equals $274.80, or $2.75 for each lb. of carbon present in the rock. About 20% of this precipitated gold, however, goes back into solution after continuous washing of the pulp, leaving a permanent loss of slightly over $2.00 for each pound of carbon.

To overcome this objectionable behaviour, experimentation was started simultaneously along the following lines:—

Chemical pre-treatment of the ore.
Chemical after-treatment of the mill tailings.
Physical methods: vacuum and heat.
Flotation concentration.

The chemical pre-treatment tests were conducted in the following manner: — Samples of pulverized graphitic schist were agitated for varying lengths of time with solutions of a series of reagents of known but varied concentration. The solution was then filtered, mixed with a pregnant cyanide solution of known value, filtered again, and the filtrate assayed. Knowing the precipitating power of the crude sample, any saving resulting from the pre-treatment was easily calculated in percentages. Table I shows the different and extreme varieties of chemicals used and their failure to solve the problem. No better success was obtained with different re-treatment methods of the tailings. Sodium sulphide solutions of various strengths and dilutions were unable to redissolve any appreciable quantities of the precipitated gold.

The vacuum tests consisted in keeping crushed samples spread in a very thin layer for a certain length of time, and then bringing them immediately in contact with pregnant solution. The table on page 114 shows the futility of this treatment.

Heating the ore up to 300 deg. C. did not have any beneficial effect; complete roasting was the only way to stop precipitation completely.

The flotation tests were very numerous, with different mixtures of oils and in several types of machine. Knowing

**TABLE I.—Chemical Pre-treatment Experiments.**

| | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Weight of sample (grms.) | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 |
| Hours of pre-treatment | Nil | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 20 | 12 | Nil |
| Chemical used for pre-treatment | Nil | $Ca(OH)_2$ | KOH | $FeCl_3$ | $Na_2SiO_3$ | NaCl | $Fe_2(SO_4)_3$ | $KClO_3$ | $CH_3OH$ | $Ca(OCl)_2$ | $H_2O$ | Nil |
| Value of pregnant solution, oz. per ton | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.74 | 0.608 | 0.608 |
| Value of same solution after contact with carbonaceous material, oz. per ton | 0.360 | 0.408 | 0.408 | 0.432 | 0.406 | 0.406 | 0.440 | 0.432 | 0.500 | 0.400 | 0.385 | 0.364 |
| Quantity of pregnant solution added (grms.) | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 |
| Total gold precipitated without pre-treatment, (oz.) | 22.8 | | | | | | | | | | | 14.64 |
| Total gold precipitated with pre-treatment, (oz.) | .... | 19.92 | 19.92 | 18.48 | 20.04 | 20.04 | 18.00 | 18.48 | 14.40 | 20.40 | 13.38 | .... |
| % of gold saved due to special treatment | .... | 12.6 | 12.6 | 18.9 | 12.1 | 12.1 | 21.0 | 18.9 | 37.0 | 10.5 | 8.6 | .. |

**TABLE I.—Chemical Pre-treatment Experiments. (continued)**

| | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Weight of sample (grms.) | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 |
| Hours of pre-treatment | 12 | 12 | 12 | 12 | 12 | 12 | Nil | 5 | Nil | 10 | 10 | 10 |
| Chemical used for pre-treatment | $4KCN\ Cu_2(CN)_2$ | $Na_2Cr_2O_7$ | $CrO_3$ | $KMnO_4$ | HCl | $H_2SO_4$ | Nil | $AgNa(CN)_2$ | Nil | $AgNa(CN)_2$ | $ZnK_2(CN)_4$ | $Pb(CH_3—CO_2)_2$ |
| Value of pregnant solution, oz. per ton | 0.608 | 0.608 | 0.608 | 0.608 | 0.584 | 0.584 | 0.644 | 0.644 | 0.632 | 0.632 | 0.632 | 0.632 |
| Value of same solution after contact with carbonaceous material, oz. per ton | 0.400 | 0.400 | 0.448 | 0.500 | 0.448 | 0.416 | 0.228 | | 0.270 | 0.340 | 0.306 | 0.352 |
| Quantity of pregnant solution added (grms.) | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 |
| Total gold precipitated without pre-treatment (oz.) | | | | | | | 24.96 | | 21.72 | | | |
| Total gold precipitated with pre-treatment, (oz.) | 12.48 | 12.48 | 9.60 | 6.48 | 8.16 | 10.08 | | 15.50 | | 17.52 | 18.24 | 16.80 |
| % of gold saved due to special treatment | 14.7 | 14.7 | 34.4 | 55.7 | | | | | | 19.3 | 16.1 | 22.6 |

TABLE II.—Vacuum Experiments.

| | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| Weight of sample (grms.) | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 |
| Vacuum subjected to | Nil | 27″ | 27″ | 27″ | 27″ | 27″ | 27″ | 27″ | 27″ |
| Mesh size of sample | —100 | +6 | +10 | +20 | +35 | +65 | +100 | —100 | —100 |
| Hours under vacuum | Nil | 5 | 5 | 5 | 5 | 5 | 5 | 5 | 5 |
| Solution taken (grms.) | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 300 |
| Gold (oz.) in solution before contact with graphitic schist | 0.456 | 0.456 | 0.456 | 0.456 | 0.456 | 0.456 | 0.456 | 0.456 | 0.456 |
| Gold (oz.) in solution after precipitation | 0.124 | 0.172 | 0.108 | 0.116 | 0.120 | 0.100 | 0.096 | 0.140 | 0.104 |
| Total gold precipitated, ounces | 19.92 | 17.24 | 20.88 | 20.40 | 20.16 | 21.36 | 21.60 | 17.76 | 20.52 |

that the harm was due to the carbonaceous matter, and that the amount of carbon varied between 5 and 10%, our method of attack was to crush the ore in water, eliminate the graphitic material as a flotation concentrate, and, in case of any values floating up with such concentrate, to run this product over tables, returning the valuable part to the flotation tails, which were to undergo the usual cyanide treatment for the extraction of the gold.

As the results of the flotation experiments led to the finding of a successful process for the treatment of carbonaceous ores, particulars of a few of these tests will be given in detail:

*Test F-1.* — A sample of ore from "graphite dump" was ground to fine pulp, and 1,000 grms. treated in flotation machine (laboratory type) for one hour, using No. 5 G.N.S. oil. A very careful skimming of the froth was made. Results:

| Product | Weight, grms. | Assay per ton | Assay x weight | % of Au | % of total weight |
|---|---|---|---|---|---|
| Concentrate | 45 | $45.60 | 2,052 | 13.7 | 4.5 |
| Tails | 955 | $13.60 | 12,988 | 86.3 | 95.5 |

The elimination of graphitic material was incomplete, since, under the microscope, one could see a great number of carbonaceous particles. A sample of the flotation tails weighing 0.5 A. T. (assay ton) was subjected to agitation with 10 A. of a $5.00 solution to determine to what extent its precipitating power was reduced by eliminating 4.5% of almost pure "graphite". No precipitation occurred. Of the $13.60, all but $1.00 went into solution in 24 hours of bottle agitation.

*Test F-3.* — A sample from "graphite dump" was treated for 45 minutes in flotation machine with ⅔ lb. (per ton) of No. 5 oil. Froth removed during the first 10 minutes was kept separate from the remainder of the froth. Results:

| Product | Weight, grms. | Assay | % Au | % of weight |
|---|---|---|---|---|
| No. 1 Froth (10 min.)..... | 15 | $22.00 | 3.6 | 1.5 |
| No. 2 Froth..... | 50 | 10.40 | 5.7 | 5.0 |
| Tails..... | 935 | 8.80 | 90.8 | 93.5 |
| | Heads by assay = $9.60 | | | |

After agitation with *barren solution* the flotation tails were reduced to $0.80.

*No precipitation, although an appreciable quantity of graphitic substance was left in tails.*

*Test F-8.* — Barren graphitic sample; precipitating power of each ton of this schist is equal to $429.60.

1,000 grms. of finely ground sample treated in flotation cell with a mixture of No. 5 and coal oil. Test was poor as machine worked badly, and large amount of oil had to be used. Results:

No. 1 Froth—150 grms. or 23.8% of total weight of sample
No. 2 Froth—280 " " 44.4% " " "
Tails —200 " " 31.8% " " "

5 grms. of the above products were agitated for 24 hours with 300 c.c. of pregnant solution assaying $10.00 per ton.

| | | |
|---|---|---|
| No. 1 Froth precipitated | $334.80 x 23.80% = | $7,968.24 |
| No. 2 Froth " | 111.60 x 44.40% = | 4,955.04 |
| Tails " | 2.40 x 31.80% = | 76.32 |
| | | 12,999.60 |

or a precipitation of only $129.99 per ton of graphitic schist, instead of $429.60; and the reduction of 70% in the precipitating power had to be credited to the action of some or all of the compounds used in the test.

*Test F-9.* — Sample of barren graphitic rock, pulverized and treated in flotation machine with coal oil and No. 5 G.N.S. Precipitating power of crude sample, $300.00 per ton.

The different flotation products were subjected to determination of precipitating power, as in the previous test:

| | | | | | | |
|---|---|---|---|---|---|---|
| No. 1 Froth | —165 grms. | or 21.7% | of total wt. | | Pptg. power = $138 = | 2,994.60 |
| No. 2 Froth | —255 " | 33.5% | " | " | " | 78.00 = 2,613.00 |
| Tails | —340 " | 44.8% | " | " | " | 4.80 = 215.00 |
| | | | | | | 5,822.60 |

or $58.22 is the precipitating power of each ton of carbonaceous schist, after it has had the beneficial treatment in the flotation cell. The original harm of the crude is reduced by 80.6%.

Having established beyond doubt that oils reduce to a great extent the precipitating properties of carbonaceous rocks, we, of course, followed our experiments along these lines.

The procedure was to pre-treat graphitic samples, omitting separation by flotation entirely, with different quantities and grades of oils, filter, and then have the carbonaceous matter act on a pregnant gold solution.

A sample of pure carbonaceous matter, with a precipitating power of 22 oz. Au, was pre-treated with different grades of crude oil or its derivatives. The effect of fuel oil or kerosene is given in table III, wherein it is shown that a reduction of 99% of the precipitating properties of this schist results.

Having found a method of neutralizing the objectionable properties of barren carbonaceous schist, the next experiments took the form of extraction tests on ore contaminated with such schist. The results of these tests are shown in table IV, and the tests prove the advantage of the oil pre-treatment.

## TABLE III—Saturation of Precipitating Power of Purely Carbonaceous Rock.

| | | | | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| PRE-TREATMENT | Weight of Sample (Grams.) | 30 | 30 | 15 | 15 | 30 | 15 | 3 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 | 30 |
| | Hours | 12 | 12 | 12 | 12 | 12 | 12 | No | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 | 3 |
| | Water (Grms.) | 290 | 290 | 115 | 115 | 290 | 115 | No | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 |
| | Kerosene (c.c.) | 5 | 0 | 0 | 0 | 0 | 5 | No | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| | Fuel Oil (c.c.) | 0 | 5 | 5 | 5 | 5 | 0 | No | 0.1 | 0.2 | 0.4 | 0.6 | 0.9 | 1.3 | 1.8 | 2.1 | 2.5 | 3.0 | 3.5 | 5 |
| | Filtration | Yes | Yes | Yes | Yes | Yes | Yes | No | No | No | No | No | No | No | No | No | No | No | No | No |
| | Grams of water left in sample | 10 | 20 | 10 | 10 | 20 | 10 | No | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| PRECIPITATION | Grams of pregnant solution added | 280 | 212 | 110 | 110 | 234 | 258 | 300 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 | 150 |
| | Value of pregnant solution per ton ($) | 4.84 | 4.82 | 3.04 | 3.04 | 4.86 | 2.78 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 | 4.79 |
| | Hours of agitation | 24 | 24 | 24 | 24 | 24 | 24 | 0.5 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 | 15 |
| | Value of solution after agitation ($) | 4.24 | 4.48 | 3.04 | 3.08 | 4.04 | 2.32 | 0.32 | Trace | 0.08 | 0.50 | 0.87 | 1.32 | 1.65 | 1.90 | 1.98 | 1.98 | 1.98 | 2.03 | 2.07 |
| | Precipitation per ton of schist, in $ | 4.06 | 0 | 0 | 0 | 3.70 | 6.23 | 447.00 | All | 23.15 | 18.95 | 15.25 | 10.75 | 7.45 | 4.95 | 4.15 | 4.15 | 4.15 | 3.65 | 3.25 |

**TABLE IV—Extraction on Graphitic Ores.**

| Weight of sample (Grms) | 90 | 90 | 60 | 60 | 60 | 60 | 60 | 60 | 60 | 60 | 60 |
|---|---|---|---|---|---|---|---|---|---|---|---|
| Water added (Grms.) | 300 | 300 | 300 | 300 | 300 | 300 | 300 | 210 | 210 | 210 | 210 |
| Fuel oil (c.c.) | Nil | 1.2 | Nil | 0.2 | 0.4 | 0.6 | 1.2 | Nil | 1 | 1 | 1 |
| Hours of treatment | Nil | 3 | 3 | 3 | 3 | 3 | 3 | 1 | 1 | 1 | 1 |
| NaCN and CaO added. Grams of each | 0.25 | 0.25 | 0.2 | 0.2 | 0.2 | 0.2 | 0. 2 | 0.2 | 0.2 | 0.2 | 0.2 |
| Hours of cyanide treatment | 43 | 43 | 41 | 38 | 38 | 38 | 38 | 36 | 36 | 36 | 36 |
| Value of solution after agitation, $ | 3.39 | 4.34 | 0.16 | 1.49 | 1.61 | 1.61 | 1.72 | 0.80 | 2.52 | 2.68 | 2.60 |
| Extraction by solution,$ | 11.18 | 14.32 | 0.80 | 7.45 | 8.05 | 8.05 | 8.60 | 2.80 | 8.82 | 9.38 | 9.10 |
| Value of tailings, $ | 4.07 | 0.83 | 8.86 | 2.06 | 2.06 | 1.86 | 1.12 | 6.60 | 1.20 | 0.60 | 0.60 |
| Composite head assay, $ | 15.25 | 15.15 | 9.66 | 9.51 | 10.11 | 9.91 | 9.72 | 9.40 | 10.02 | 9.98 | 9.70 |
| Remarks | Graphitic ore | | Strongly graphitic | | | | | Strongly graphitic | | With approximately 2% Carbonaceous ore | |

## FIRST MILL RUN WITH OIL PRE-TREATMENT.

We hope in the near future to be able to give details of the actual milling operations, using the oil pre-treatment process which overcomes the objectionable features of carbonaceous rock in a gold cyanide solution. For the present, a brief description of the first mill run on graphitic ore will be given.

The character of the ore milled was about 20% "graphitic", and it contained some big barren slabs of pure carbonaceous schist. To determine the possible extraction by direct cyanide treatment, a portion of the treated sample, ground to the same fineness as in our mill practice, was agitated for several days in a cyanide solution. The value of the tailings remained at $3.00.

The crushing and grinding of the ore was in water, and the pre-treatment agent, a mixture of crude fuel and gas oil was added directly to the ball mill and tube mill scoops. The oil

was added by a very small but continuous flow from an inverted 5-gallon can, fitted with special cock. The apparatus was crude, so that the amount of oil added could not be regulated; but its consumption per ton of ore at the end of the mill run equalled 0.76 gallons.

The finished pulp from the grinding circuit flowed directly to a Dorr agitator (No. 1). No water was decanted or filtered out. The cyanide and lime was added directly to the agitating tank, and the solution made up to usual strength, which is about one pound of each chemical per ton of water. The solution and pulp samples in the agitator were taken every hour for the first 25 hours of agitation, and then every 2 hours. A full pail of pulp taken each time was allowed to settle and then filtered. The filtrate was collected in two quart bottles, one of which was kept for our assay office and the duplicate for "umpire". The sand samples were at first taken every 3 hours, and finally every 5 hours, at the end of the distributing arms of the Dorr mechanism.

The consumption of cyanide was the same as in our direct cyanide practice, and no mechanical troubles, as frothing, foaming, etc., occurred due to the addition of oil.

Table V gives the complete data of this mill test. In 24 hours $12.60 were extracted from a $13.60 carbonaceous ore. This mill test corroborates the long list of earlier experiments as to the efficacy of the above-mentioned oils as an agent for neutralizing and destroying the precipitating properties of a carbonaceous schist on a gold cyanide solution.

In conclusion, the writer takes this opportunity of acknowledging his indebtedness to Mr. R. J. Ennis, general manager, whose advice and active interest throughout the course of these experiments and tests has contributed greatly to the successful results obtained.

## TABLE V.—Mill Run of Carbonaceous Ore

**Assay of Heads — Sample No. 0 = $13.60**

| No. of sample | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Hours of agitation | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 |
| KCN Pounds per ton | 1.45 | 1.5 | 1.45 | 1.45 | 1.4 | 1.45 | 1.45 | 1.45 | 1.40 | 1.4 | 1.35 | 1.4 | 1.4 | 1.35 | 35 | 1.4 |
| CaO Pounds per ton | .65 | .7 | .7 | .7 | .7 | .65 | .65 | .6 | .65 | .6 | .6 | .6 | .65 | .65 | .65 | .6 |
| Value per ton of solution, $ | 1.68 | 2.40 | 2.68 | 2.92 | 3.04 | 3.12 | 3.16 | 3.20 | 3.20 | 3.32 | 3.48 | 3.44 | 3.40 | 3.20 | 3.40 | 3.24 |
| Extraction per ton of ore by solution assay ratio 3.842, $ | 6.45 | 9.22 | 10.29 | 11.21 | 11.67 | 11.98 | 12.14 | 12.29 | 12.29 | 12.75 | 13.37 | 13.21 | 13.06 | 12.29 | 13.06 | 12.44 |
| No. of corresponding tail sample | | A | | B | | | C | | | D | | | E | | | F |
| Value of tails, $ | | 4.10 | | 3.80 | | | 2.40 | | | 2.40 | | | 1.60 | | | 1.60 |
| Umpire solution assay, $ | 1.49 | 2.16 | 2.47 | 2.65 | 2.79 | 2.72 | 2.99 | 3.04 | 3.11 | 3.11 | 3.16 | 3.19 | 3.14 | 3.21 | 3.21 | 3.21 |

## TABLE V.—Mill Run of Carbonaceous Ore (Continued)

Assay of Heads—Sample No. 0. = $13.60

| | | | | | | | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| No. of sample | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 |
| Hours of agitation | 18 | 19 | 20 | 21 | 22 | 23 | 25 | 27 | 29 | 31 | 33 | 35 | 37 | 39 | 41 | 44 | 62 |
| KCN Pounds per ton | 1.4 | 1.4 | 1.35 | 1.35 | 1.4 | 1 3 | 1.3 | 1.3 | 1.35 | 1.3 | 1.35 | 1 3 | 1.3 | 1.35 | 1.3 | 1.25 | 1.25 |
| CaO Pounds per ton | .6 | .55 | .55 | .5 | .5 | .55 | .55 | .55 | .5 | .5 | .5 | .5 | .45 | .5 | .5 | .45 | .4 |
| Value per ton of solution, $ | 3.48 | 3.72 | 3.16 | 3.16 | 3.16 | 3.28 | 3.28 | 3.28 | 3.28 | 3.28 | 3.28 | 3.28 | 3.44 | 3.28 | 3.36 | 3.28 | 3.40 |
| Extraction per ton of ore by solution assay ratio 3.842, $ | 13.37 | 14.29 | 12.14 | 12.14 | 12.14 | 12.60 | 12.60 | 12.60 | 12.60 | 12.60 | 12.60 | 12.60 | 12.79 | 12.60 | 12.90 | 12.60 | 13.06 |
| No. of corresponding tail sample | | | G | | | | H | | | I | | J | | | | K | |
| Value of tails, $ | | | 1.60 | | | | 1.00 | | | 1 00 | | 1.00 | | | | 1.00 | |
| Umpire solution assay, $ | 3.24 | 3.25 | 3.30 | 3.35 | 3.34 | 3.33 | 3.38 | 3.43 | 3.45 | 3.45 | 3.48 | 3.49 | 3.50 | 3.53 | 3.50 | 3.45 | |

# THE RECOVERY OF PRECIOUS METALS FROM SUDBURY ORES.

By R. L. Peek.

Annual General Meeting, Ottawa, March, 1922.

The nickel-copper ores of the Sudbury district have long been known to carry small amounts of gold, silver, and metals of the platinum group. The amount of these metals is usually very small, but in some ore-bodies their value has at times exceeded that of the nickel content of the ore. There is good reason for saying "*at times*", because in the past thirty years the price of nickel has varied only from about twenty-four to forty-five cents per pound, whereas, in the same period, platinum and palladium have exhibited a remarkable range, the former from about $15 to $175, and the latter from $7 to $160, per troy ounce.

The so-called "off-set" ore-bodies appear to have the highest relative content of precious metals. Attempts have been made from time to time to determine the associations of the minerals carrying these metals, but very little has been published to shed light on the subject. The following notes may therefore be of interest.

A quantity of ore from the Murray mine was painstakingly and laboriously sorted into the following classes:—

No. 1 Clean pyrrhotite.
No. 2 Pyrrhotite with small amounts of chalcopyrite.
No. 3 Pyrrhotite with much chalcopyrite.
No. 4 Fine screenings.

Assays of these yielded the following results:—

| | Per cent | | | | Grms. per ton | | |
|---|---|---|---|---|---|---|---|
| | Ni | Cu | Fe | S | Au | Pt | Pd |
| No. 1..... | 3.45 | Trace | 48.0 | 31.35 | 0.269 | 0.463 | 0.680 |
| No. 2..... | 3.00 | 0.10 | 45.2 | 29.23 | 0.132 | 0.408 | 0.605 |
| No. 3..... | 1.90 | 1.00 | 32.8 | 18.45 | 0.170 | 0.399 | 0.574 |
| No. 4..... | 2.74 | 0.44 | 38.6 | 29.90 | 0.115 | 0.436 | 0.634 |

Another set of samples was collected by sorting and cobbing ore as raised from the mine. Assays of these were as follows:—

No. 1 Smelting ore.
No. 2 Reject.
No. 3 Pyrrhotite.
No. 4 Chalcopyrite.

| | Per cent | | | | Grms. per ton | | | |
|---|---|---|---|---|---|---|---|---|
| | Ni | Cu | Fe | S | Au | Pt | Pd | Ir |
| No. 1.... | 1.47 | 1.17 | 30.78 | 15.46 | 0.12 | 0.292 | 0.416 | 0.034 |
| No. 2.... | 0.29 | 0.85 | 14.15 | 3.53 | 0.05 | 0.101 | 0.161 | 0.0105 |
| No. 3.... | 3.82 | Trace | 55.25 | 34.79 | 0.15 | 0.402 | 0.584 | 0.064 |
| No. 4.... | 0.19 | 16.35 | 18.10 | 17.66 | 0.44 | 0.424 | 0.608 | 0.084 |

A certain amount of amusement may be had by determining ratios between these minute quantities of the several precious metals present in the ore and the nickel, copper, iron and sulphur content, singly and in various combinations. The only useful result of such gymnastics is that support is apparently lent to the commonly held impression that the relatively low-grade ores are apt to run proportionately high in precious metals.

It is to be noted that, for platinum group assays on the Sudbury ores, it is customary to use portions weighing many kilos. This makes the assay very slow and expensive, so it is not remarkable that there are relatively few made.

On account of the very large tonnages of ore smelted, the aggregate of the precious metals present in the matte produced has been very considerable. If one will refer to that excellent encyclopaedia of the nickel industry, the Report of the Royal Ontario Nickel Commission, he will find on page 486 some records of assays on roasted matte, showing the content therein of silver, gold, platinum, palladium, and iridium. In each case, also, the percentage of nickel and of copper in the matte before roasting is given. If we assume that the matte was roasted substantially free from sulphur, and re-calculate the assays to a basis of un-roasted matte, we get:—

Troy ounces per ton of 2,000 lb.

| | Sample No. 1 | | Sample No. 2 | |
|---|---|---|---|---|
| | Roasted Matte | Un-roasted Matte | Roasted Matte | Un-roasted Matte |
| Silver | 1.84 | 3.32 | 6.77 | 16.47 |
| Gold | 0.027 | 0.049 | 0.282 | 0.686 |
| Platinum | 0.1235 | 0.223 | 1.087 | 2.645 |
| Palladium | 0.197 | 0.356 | 1.082 | 2.632 |
| Iridium | 0.046 | 0.083 | 0.072 | 0.175 |

From returns published in the reports of the Ontario Bureau of Mines and of the Mines Branch of the Federal Department of Mines, we may conclude that the production, equivalent to bessemer matte, of the two companies whose matte was assayed for the Nickel Commission, was, to the end of 1920, approximately as follows:— Company No. 1, 703,500 tons, and Company No. 2, 147,100 tons. On the basis of the above assays and these tonnages, the production of precious metals in matte from Sudbury ores by the two major operating companies to the end of 1920 would be:—

| | No. 1 | No. 2 | Total |
|---|---|---|---|
| Silver, Troy oz. | 1,294,900 | 995,900 | 2,290,800 |
| Gold, " | 19,000 | 41,400 | 60,400 |
| Platinum, " | 86,900 | 159,900 | 246,800 |
| Palladium, " | 138,600 | 156,700 | 295,300 |
| Iridium " | 32,300 | 10,500 | 42,800 |

While the basis for this estimate is justly subject to criticism, it may be said, without going into details, that it is actually a very fair approximation.

Company No. 1 recovers a relatively small proportion of the platinum metals, as their process does not readily lend itself to high recoveries. Company No. 2 should not sustain any serious loss of the platinum metals in the course of the nickel-copper separation.

When the ores are smelted, substantially the whole of their precious metal content enters the matte. Relatively

to the other metals, their total amount is so low that it is unlikely that any loss of precious metals occurs in bessemerizing, although iridium and osmium are very prone to oxidize and enter slags or volatilize. In the treatment of the matte for separation of nickel and copper, there is opportunity to economically recover the precious metals in a concentrate which admits of treatment for their separation and refining. When the matte is used for making monel metal, however, there is probably no recovery of the precious metals.

In the Orford process for separating nickel and copper by fusion with alkaline sulphide and gravity separation of the nickel sulphide from copper-sodium sulphide, it is found that by far the greater part of the platinum group metals accompany the nickel sulphide, while gold and silver tend to concentrate in the copper-sodium tops. The tops are blown up to blister copper, which is electrolytically refined, and the silver and gold are recovered in the anode slimes. Some platinum and a lesser amount of palladium is also recovered here, but by far the larger part is found in the nickel bottoms.

The nickel bottoms are ground up and treated with dilute sulphuric acid, whereby nearly all the iron, remaining after the smelting operation, is dissolved and leached out. The matte is then roasted at a low heat, and after the bulk of the sulphur has been burned off and most of the metals oxidized, niter cake and salt are thoroughly mixed in and the material banked high and kept at a suitable temperature for several hours to chloridize the copper. This "first calcine" is leached with dilute acid, whereby most of the copper is leached out. It is again roasted with niter cake and the leaching repeated, until the copper is reduced to the desired extent. The leach liquors from the first and second calcinations (but especially the first), contain some of the platinum group metals in solution, and by slowly flowing these liquors over scrap iron, the precious metals cement out with the copper.

The cement copper is, or at least was, fire refined and shotted, and used for making bluestone (copper sulphate), and in this operation the platinum metals collected in the sludge

from the vitriolization towers. The platinum-bearing sludge is worked up in the platinum refinery at the Orford works. It was quite impressive in the good old days to see quantities of palladium stored in a cupboard patiently awaiting sale to some manufacturer of dental supplies, who did not want as much palladium as could be produced, and who was only prepared to pay about $6 per ounce for it. Platinum at that time was worth a mere $16 per ounce.

Having in view the difficulty and expense of extracting more of the precious metals than would come out incidentally to the purifying of the nickel oxide, many experiments were made with the object of discovering a way of treating the matte so as to obtain a better recovery. When the Mond process was worked out there appeared a good opportunity to recover substantially all the precious metals contained in the matte to be treated by that process. Recognition of this fact would naturally lead to the acquisition of ore deposits relatively high in precious metal content.

Provided chlorides are absent, the platinum metals are only very slightly soluble in sulphuric acid or acid solutions of copper and nickel sulphates. A rough idea of the losses to be expected when leaching copper out of roasted matte may be gained from the following figures: for every 1,000,000 parts copper dissolved, there is dissolved or carried out in suspension

| | |
|---|---|
| Silver | 30 parts |
| Gold | 0.3 parts |
| Platinum | 0.4-0.5 parts |
| Palladium | 0.5-0.7 parts |
| Iridium | 0.7 parts. |

It will be seen that iridium is relatively much more soluble — it may even be five or ten times more soluble — than platinum or palladium. Possibly these losses would be somewhat reduced by using more efficient filtering and clarifying arrangements. The presence of chlorides, of course, increases the losses.

There probably are no measurable losses in the volatilizers, so the Mond process should recover substantially all the precious metals in the final residue from the matte treatment.

The Hybinette electrolytic process, like the electrolytic copper refining, concentrates practically the whole of the precious metals in anode slimes. Owing to the high sulphur content of the anodes, the slimes contain very large amounts of sulphur. This, however, is very easily burned off, leaving a residue which is easily concentrated to the point where separation of the platinum metals may be undertaken.

The usual way for working the residues or slimes for separation of the precious metals is to first roast them to get rid of most of the sulphur and to oxidize the copper. The bulk of the copper is then removed by leaching with dilute sulphuric acid, any palladium or silver going into solution being later cemented out on copper. Part of the nickel is also dissolved. The insoluble portion, which now contains from 8 to 10% platinum metals, is fused with lead and slag-forming materials, such as lime and sand, whereby the base metals and non-metallic components of the residues are for the most part slagged. Losses of the precious metals in the slag are trifling, though iridium is slagged proportionally more than any of the others. The lead bullion is cupelled with so much silver as to give an alloy of about 15 to 20 parts silver to one part of the other precious metals. The cupellation is sometimes finished under a stream of oxygen, in order to raise the temperature and promote the separation of as much base metal as possible. After cupellation, the metal is either cast as anodes or granulated. The litharges always contain much iridium and silver and lesser amounts of the other metals, and are sent back to the first smelting.

The bullion, now containing about 93 to 96% silver, and the balance platinum metals and gold, is electrolysed or dissolved in sulphuric acid. In either case a substantial part of the palladium goes into solution, but practically none of the platinum. If electrolysis is used, measures have to be taken to prevent palladium from plating out with silver.

The residue remaining from the silver separation consists of part of the palladium and all of the platinum, iridium, rhodium, ruthenium, and gold. The residue still contains a little silver also. About now the real trouble arises. The separation reactions and methods were worked out long ago and published,

mostly in 1859 and the '70's, but the details of the procedure now employed by various works are jealously guarded, probably to the mutual disadvantage of all concerned.

In general, it may be said that the whole of the palladium that is dissolved with the silver is easily and cheaply recovered in a pure state, as there is only a little rhodium to be taken care of. The mixture of metals not dissolved by the sulphuric acid is usually first treated with repeated doses of nitro-hydrochloric acid, whereby the greater part of the platinum, palladium and rhodium, and a small part of the iridium and ruthenium, is dissolved. The portion remaining insoluble after this treatment is usually fused with sodium peroxide to form soluble sodium ruthenate and osmate. Unhappily, part of the iridium is also oxidized and so goes into solution when the soda-fusion is dissolved. It is cemented out on zinc after the osmium and ruthenium have been boiled off in a current of chlorine. It may be taken as a general rule that each member of the platinum group seems to catalyse to some extent every possible reaction that the other members present may possibly participate in. The result is that, in separating these metals, the proceedings resolve themselves into a succession of approximate separations and subsequent purifications of the metals so separated.

The chloride solution, containing substantially all the platinum, is twice evaporated to dryness with an excess of hydrochloric acid, to get rid of the nitric acid, and the platinum, iridium, and a part of the rhodium are then precipitated with ammonium chloride. Palladium is recovered as potassium chloro-palladite from the filtrate.

To go into all the details of the separations and purification of the several metals would be burdensome. To give some idea how difficult control work is on the process, the following ratios between the metals are given. These ratios were obtained by two of the largest refineries treating platinum metals. In the table below the figures have been re-calculated from those originally given so as to bring them to a common basis for the sake of easier comparison.

| | No. 1 | No. 2 |
|---|---|---|
| Platinum | 1.000 | 1.000 |
| Palladium | 0.747 | 1.491 |
| Iridium | 0.083 | 0.049 |
| Rhodium | 0.117 | Trace |
| Ruthenium | 0.059 | ? |

Certain interesting peculiarities of the platinum metals and their compounds may be mentioned. Palladium exhibits a propensity to sinter at very low temperatures relative to its melting point. A voluminous mass of palladium sponge will, if heated to a dull red in the absence of oxygen, sinter into a hard compact mass in an hour or so, giving almost the impression that some of it must have been taken away.

Platinum in a sulphate solution readily forms with ammonia the very dangerous compound $NH_3Pt_2(OH)_{10}$. This substance detonates violently at less than a red heat. On one occasion about two grams of the compound was placed in a roasting dish and put in a large muffle to dry. It detonated, entirely destroying the muffle, and, incidentally, temporarily deafened the unhappy man who stood near.

It should not be overlooked that, owing to the invention of cheap substitutes for platinum for use in the arts, and to the caprices of fashion, which affect the use of platinum in jewellery, the production of this metal and its allies may be classed as hazardous in a commercial sense, except where it may be recovered as a by-product, as in the case of the Sudbury nickel ores.

## DISCUSSION.

DR. ALFRED STANSFIELD: A paper like this by Mr. Peek, dealing mainly with the metallurgy of the platinum metals, is very useful at the present time in view of the great demand for these metals and of the fact that platinum is found in British Columbia.

The figures in the table on page 124 show that platinum metals having a present value of some millions of dollars have been smelted from Sudbury ore, but no doubt the greater part of these metals have been wasted; some being perhaps in the form of nickel steel sunk by the Germans at Scapa Flow.

I understand that figures have been published showing the amount of platinum metals that have been recovered from the Sudbury ores. It would be interesting to compare these with the figures given by Mr. Peek.

A comparison of the values of the metals in the mattes referred to on page 124 shows at recent market quotations[1] that the gold and silver are of small value, being less than $2.00 in sample No. 1, and nearly $10.00 in sample No. 2, per ton of matte. The platinum metals are worth $30 in No. 1 and $170 in No. 2, while the nickel and copper (the percentages of which are given in the Ontario Nickel Report) are worth $500 in No. 1 and $450 in No. 2. It will be seen then that the platinum metals represent 5% of the value of sample No. 1, and 26% of the value of sample No. 2. In the first case the metals are worth saving as a by-product if this can be done without serious expense, while in the second case they form, at present prices, an important part of the commercial product.

It must be remembered that these figures represent the value obtainable if all the metals in the matte could be recovered, and without making any deduction for the cost of recovery and refining. These considerations, however, apply to the nickel and copper as well as to the platinum metals and so the simple comparison I have made has a certain value.

In reading this paper I was puzzled by the great difference between the figures representing the precious metals in the "roasted" and "unroasted" matte given in the table on page 124. I compared the figures and found for sample No. 1 a constant ratio of 1.80 and for sample No. 2 a constant ratio of 2.43, thus indicating that the roasted mattes weighed respectively 1.80 times and 2.43 times the raw matte. A consideration of the probable analyses of these mattes both raw and roasted, assuming the roasted matte to consist of the oxides CuO, NiO, and $Fe_3O_4$ gave ratios of 1.03 and 1.10, respectively, for the two samples, so that apparently some mistake has been made.

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(1) Feb. 15th, 1922.
Nickel..... 40 cents per pound.
Copper..... 13 cents per pound
Platinum..... $90 per ounce.
Iridium.....$180 per ounce.
Palladium. . .. $60 per ounce.

Comparing the figures under "roasted matte" with those in the Nickel Commission report, I found that while the figures for sample No. 1 had been correctly reported, the figures for sample No. 2 had all been raised 10%. It appeared from this that the figures headed "Roasted Matte" were probably intended to indicate the contents of the *raw* matte, as in sample No. 1 the raw and roasted matte were almost identical in weight, while in sample No. 2 there is a difference of 10% in the weights.

The columns marked "Roasted Matte" must then have some other meaning, and, remembering that sample No. 1 contains 55.4% of nickel (= 100 ÷ 1.80) and sample No. 2 contains 41.1% nickel (= 100 ÷ 2.43), it was clear that these columns indicate the precious metals per ton of the metallic nickel in the matte.

I have retailed this little investigation at length as it forms an amusing example of what may be called by analogy the "Higher Criticism" in metallurgy. It does not at all affect Mr. Peek's conclusions, as he has used the correct data in making up the table.

In conclusion I should be glad to have a little additional explanation of the table on page 129. Am I correct in assuming that No. 1 and No. 2 are assay values of the platinum metals in duplicate samples sent to two refineries, the assays being both reduced proportionally so as to make the platinum in each equal to unity — in which case it would also be interesting to find how nearly they agreed in the platinum — or do they represent the amounts of the platinum metals reported or actually recovered in the treatment of considerable quantities of platinum material from one source but treated by two refineries?

PROF. GEO. A. GUESS: Mr. Peek's paper contains some very interesting figures and information. That there has not been a more complete recovery of platinum and its associated metals from our Canadian nickel ores is, I think, a matter of regret to everyone. At the present prices, the precious metal content of matte No. 1 is $33.32 per ton, which will go a long way towards paying refining charges. The figures grouped under No. 1 show a Pt + Pd total of 225,500 oz. It is altogether probable that the correct total is not known. We do know that in the days of the operation of the old Copper Cliff

mine the matte product was very much higher in these metals than at present, probably well over an ounce per ton of matte for the combined metals. I have been led to believe that the Pt + Pd content in later times was from 0.35 to 0.37 oz. per ton of matte, which is slightly higher than the figure taken by Mr. Peek for the whole output.

The Orford process cannot in these days be called a refining process. Those on the outside consider it is not on the same high plane of efficiency as the other operations of the company. It is a process for separating nickel and copper with only a slight recovery possible of the rare metals. When monel metal is made there is no recovery whatever of the platinum metals.

To create a demand for nickel by increasing its use is, I think, a distinct duty for Canadian metallurgists and manufacturers in co-operation with the producers.

Undoubtedly the Aluminium Company of America has shown how this may be done. Nearly every kitchen throughout the country is provided with aluminium pots and pans. We know how much better nickel would serve for this purpose.

The work of Mr. C. P. Madsen in making pliable nickel electrolytically, of which Mr. Peek could tell us a good deal if he chose, may make it possible to produce a material more easily prepared for these markets. This in turn may lead to the proper refining of the nickel and to the recovery of the platinum metals.

Since Russia ceased producing platinum, Colombia has been the world's chief source of the metal, with a production of about 30,000 oz. annually. One of the largest concerns interested in Colombia is the General Development Company — the Lewisohns of New York. The Colombian output has more than doubled in the last ten years.

The metal platinum was first discovered in about the area where it is still being mined in Colombia, i.e., the Choco district. There have been recovered from graves in Ecquador many specimens of ornamental jewellery made of platinum, often associated with gold. These are said to be about 2,000 years old, and they show that the modern craving for platinum is nothing new.

It was the Spaniards who first named the metal, the word *platina* being a diminutive form of *plata*, the Spanish for silver. The earliest knowledge possessed by Europeans concerning

platinum dates from 1735, when an expedition under the direction of the Royal Academy of Paris was sent to South America to measure the arc of a meridian. The account of this expedition, written by Don Antonio de Ulloa, a Spaniard, was not published until 1748. He says: "In the Choco district is much placer mining in which the gold is so disguised and mixed with other metallic substances, juices, and stones, that for their separation from the gold they are obliged to use quicksilver. Sometimes they find mineral substances where the platina (a stone of such resistance that it is not easy to break it nor reduce it in size, with the force of a blow on an anvil of steel) is the cause of abandonment. It is not possible to subdue it by calcination nor is there a way to extract the metal which is encountered without expense and much labour and cost." He states further that the Indians had in places of worship mirrors wrought out of stones. His description of these is such that it is altogether probable that they were made of platinum.

The *Philosophical Transactions* of the Royal Society for 1750 contain an extract from a letter to Wm. Watson in that year, giving an account of the semi-metal "*platina del Pinto*" which he had obtained nine years previously from Mr. Chas. Wood, a metallurgist who met with it in Jamaica whither it had been brought from Carthagena in New Spain where it was also known as *Juan Blanco* ("White Jack"). He describes it as bright, shining, of uniform texture — taking a fine polish — and incapable of being extended under the hammer. He says: "It had been kept in a furnace for two hours in a heat that would run down cast iron in fifteen minutes, which great heat it endured without being melted or wasted." "Having been melted in an assay furnace on a test of lead and exposed to a great fire for three hours till the lead was wrought off, the platina was afterwards found remaining at the bottom of the test without having suffered any alteration or diminution by this operation."

The metal was also described in 1751 by Scheffer, of the Royal Academy of Sweden, under the name of "white gold."

The *Philosophical Transactions* of the Royal Society for 1754 contain a lengthy paper by Dr. Wm. Lewis covering his investigation of the chemical properties of platina. This was followed by a second paper in 1757. The papers disclose the result of a long research on the part of the author. He

discovered its solubility in aqua regia, and described the salt platinic chloride. He also succeeded in purifying it to some extent by fire methods, but his platinum had a lower gravity than gold.

This early work of Dr. Lewis on the chemistry and metallurgy of platinum was undoubtedly known to Chabaneau, to whom is probably due the credit of being the first to purify and forge the metal.

Chabaneau, who was born in 1754 in the south-west of France, studied physics and chemistry at the time of Lavoisier's teaching in France. In 1774 he became Director of a college founded at Bergara in northern Spain. After three years here he was called by Charles III of Spain to Madrid, where there was created for him a chair of physics, chemistry, and mineralory. At this period Spanish America was sending to the mint at Madrid ingots of gold and silver, and from time to time a mineral in the form of little white metallic grains, infusible and very heavy, called platina. Chabaneau undertook the difficult task of obtaining platina in metallic ingots. It had been observed previously that if metallic arsenic was added the melting point of platinum was lowered. The platinum, however, was brittle, but by heating for a long time at a high temperature the arsenic was gradually volatilized leaving the platinum malleable.

Among those who had interested themselves in the founding of the college at Bergara was the Marquess of Aranda, a man distinguished among all the nobles for his devotion to science. He held Chabaneau in high esteem, and through him the Government turned over its whole supply of platinum to Chabaneau. He furnished him with everything in his power for the undertaking. The platinum as received contained only about 77% platinum, and the difficulties he encountered in purifying it were indeed great. We are told that, on visiting the laboratory on one occasion, the Marquess found Chabaneau in a frenzy engaged in throwing out of the door and windows his dishes, flasks, and ores, as well as the solutions he had prepared with so much difficulty. "Away with it all. I'll smash the whole business," he cried. "You shall never again get me in touch with the damned metal." According to Chabaneau's biographer, lime does not precipitate platinum in artificial light, but in daylight the metal is completely

precipitated. He says that Chabaneau, working with lime at night, had been able to precipitate all the other metals present in his solution, while the platinum was left unprecipitated and purified. Repeating the operation by day, platinum and all were thrown down and he was completely at sea.[1] Three months later, however, at the home of the Marquess of Aranda, there appeared upon a table an ingot in the form of a four-inch cube, and with a beautiful lustre. It was malleable platinum. The enthusiastic Marquess started to pick it up but failed to move it. "You are joking," said he to Chabaneau. "You have fastened it down." It was no wonder he thought so, for such a mass would weigh fifty pounds.

Chabaneau's discovery consisted of compressing the platinum sponge while hot at the moment of its formation and hammering it at a white heat. The King of Spain had a commemorative medal struck in platinum and gave Chabaneau a life pension. This medal bore the date of 1783.

In 1859 Deville and Debray perfected the method of fusing platinum in the flame of the oxy-hydrogen blowpipe, but this method was later abandoned for the compression method used at present, the honour for the discovery of which belongs to Chabaneau.

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(1) Chabaneau's biographer appears to have this reaction reversed. According to some tests I have had made, lime will not precipitate platinum in daylight but will do so by artificial light or in the dark. I find no other reference to this reaction. It is a point our chemists might clear up.

# PROSPECTING CONDITIONS IN NORTHERN MANITOBA.

By R. C. Wallace.

(Annual General Meeting, Montreal, March, 1922.)

The purpose of this paper is to express the point of view of the prospector in Northern Manitoba. To what extent the prospector may be mistaken in his attitude on any point raised in this paper is, as far as the present purpose is concerned, another matter. However, it is important that the judgment of the prospector on conditions as he meets them be known; for the development of the country is, to a very large extent, in his hands, and only to the degree that he is satisfied with conditions will he remain in the country and attract others to it. An endeavour will here be made to present an accurate interpretation of the things that mining men in the northern camps say when they meet. If anywhere the true note is missed, the writer trusts that his northern friends will without hesitation put him right.

## Prospecting Ground.

The pessimist with reference to the possibilities of the mineral areas in Manitoba is a *rara avis*. Both in the copper and in the gold areas, prospectors feel that much has been accomplished in a few years, and that the ground is good enough to justify holding on for better times. Few have made money and fewer have seen their properties developed into successful mines. For this there are reasons unconnected with the potentialities of the prospects; but even under present conditions, the most capable prospectors have had little cause for complaint. There is a general feeling that prospecting for copper must be postponed till the market improves, and attention be directed to the search for gold. The Elbow lake, Herb lake, and Manigotagan areas attract the greatest numbers, though the accessibility of the Bear river and Bird river copper-nickel districts has made these fields attractive, even when the base metals are in the doldrums. There is a definite tendency

to resist the temptation — always present — to seek new fields far from transportation. The prospector has come to the conclusion that, in a province where railway transportation will not be provided to many mining fields for some time to come, it will pay to devote attention exclusively to those districts where there is a reasonable likelihood that railway facilities will be available. Where prospectors are all too few it is wise policy to concentrate on one or two districts, and develop to the stage where transportation facilities may reasonably be demanded. Outlying fields, such as the Island lake and Knee lake fields, will probably be left to the next generation to develop.

## MINING REGULATIONS.

The general feeling is that the Federal regulations are, with certain notable exceptions, fair to the prospector. There is no license fee. The work to be done in staking a claim, and holding a claim when staked, is the minimum consistent with reasonable development. The better class men, indeed, who are looking to the future of the districts, would welcome a visit from the Federal mining inspector, who is an absolutely unknown official in these parts, if only for the reason that he might insist that the lines be cut as the regulations lay down, and assure himself that the assessment work is carried out according to affidavit. It is the part of human nature to become slack if there is no checking up; and any district where it would take weeks to unravel what ground is staked, and what is free, cannot go ahead.

The great majority of prospectors are opposed, and irrevocably opposed, to the leasing system which the Federal regulations lay down. They feel that, more than anything else, this system is the stumbling block in the path of capital desirous of entering the Manitoba fields. Capital, they consider, has no faith in Governments, and the "may" of a Minister with reference to granting a renewal of a lease might well become a "will not" if his personal friends might benefit thereby. They argue that Ontario and British Columbia have had a successful history under the freehold system, and that a similar success would be good enough for Manitoba.

This is undoubtedly the most fundamental objection that our prospectors have to Federal regulations, beyond the fact that they are regulations and may be changed overnight. They feel, however, that the man who has a leasehold on his claim has not sufficient security as to surface rights, including timber, and that a large company operating near him may well secure his land and timber both; and also that the recent changes in the regulations did not help much in this regard.

## DEVELOPING THE PROSPECT.

Few of our prospectors have sufficient means to develop their own prospects. They must, therefore, deal with mining companies through their mining engineers, or assist in the organizing of a stock company to finance and operate the prospect. Between the average prospector and the average engineer there is no love lost. The engineer considers the prospector an unbalanced optimist who thinks of millions when the surface indications spell thousands, and who demands cash payments for the mere privilege of examining his property. The prospector suspects the engineer of decrying the property to the owner and praising it to the mining company. He feels that neither the engineer nor his company realize that, while their examinations and negotiations are pending, the prospector must live. As to price, he deplores the fact that there is no standard which may serve even as the roughest guide in arriving at a fair price to ask for his property and he feels that the engineer has not yet reached the stage of appreciating the fact that the hardest thing in the making of a mine is the finding of it. Because of this difference in attitude, difficulties frequently arise, and the prospector is forced back on the stock subscription system, with the inducement of better terms, and a more generous first payment. In the last year or two, however, the prospector seems to have realized that this latter system has been unsuccessful in Manitoba territory; and there has been a much more conciliatory attitude shown to engineers who represent sound mining companies, in order that the properties be given a real chance to develop into mines. This is one of the most hopeful signs of the times in our territory.

But representatives of mining companies do not visit a district unless the companies are interested in the district. The most urgent problem for the prospector and the mining man generally in this province has been the question of ways and means to elicit an intelligent and sympathetic interest on the part of mining capital in the various districts in the province. There are many things that assist, there are many things that hinder, in this work. Speaking generally, the prospectors feel that they must carry on in their work in such a way as to gain the confidence for their field of the people that matter financially. They also feel that the results accomplished in this direction during the past few years have been very real, and will bear fruit when financial conditions improve.

## GENERAL CONDITIONS.

In districts where geological maps have been prepared by the officers of the Geological Survey of Canada, as in the Flin-Flon and Herb lake areas, such maps have been very fully utilized and much appreciated by the prospector. They have led to more careful and intelligent prospecting, and a much more complete knowledge of the geological situation. There is need for equally good maps of the Manigotagan, Bear, and Bird River areas, and the work for such maps is now, I understand, in progress. The courses of instruction given from year to year at the University of Manitoba, and to some extent at The Pas, by the staff of the Geology Department of the University, have been fully taken advantage of by all who were in reach of such courses, and have undoubtedly stimulated the prospector. As the mining camps grow, the place of instruction will be moved from the towns to the camps. On the whole, the authorities of the province have done what they can to assist the industry in building summer roads and cutting winter roads to the camps. Railway transportation must await substantial production. To one of the camps (Herb lake), the telephone line now extends. To all the others, means of communication is as yet unprovided for, and is much needed. There seems little doubt that the wireless telephone will finally be adopted as the most suitable means of communication with the outlying camps.

While prospectors are much at home in the camps, they feel somewhat homeless in the city. There is a need, to some extent in Winnipeg, perhaps to a greater extent at The Pas, of club rooms for prospectors and mining men, where the current mining journals may be consulted, the most recent maps and reports made available, a complete collection of ores, rocks, and minerals of the districts set up for reference, and, in particular, where the prospector may feel at home and meet his friends. Such rooms might serve as well as the headquarters of the local branch of the C.I.M.M., to the advantage both of the prospector and the Institute. The financing of the project could not be met by club membership fees alone, either in Winnipeg or The Pas, under present conditions. If the plan were seriously pressed by the prospectors, however, there may well be public spirited bodies or individuals who will realize that an endowment for the purposes of such a club would be an investment of prime importance in the development of the industry, and who will be prepared to assist the mining men in setting up a fund for the purpose. While only a few prospectors have yet discussed this matter with the writer, if the feeling is sufficiently widespread among the mining men, the idea can doubtless be successfully put into effect.

## DISCUSSION.

MR. E. L. BRUCE: The difficulties and discouragements of the prospector in Northern Manitoba have been very clearly and fairly set forth by Dr. Wallace, and some remedies have been suggested for the handicap under which the mining industry must labour. There are one or two points that have not been particularly emphasized in his paper. He referred to the mining regulations, and to some of the weaknesses in the present laws. There are others even more serious which he has not mentioned. One of these is, in the opinion of some at least, the present Dominion regulation governing the staking of claims. As intimated in the paper, the requirements of that law regarding the proper cutting out of the location line are not always carefully observed, but even when the work is carefully performed the system is bound to be much less satisfactory than is

that of cutting out the four claim boundaries as required by the Ontario law.

The following objections may be raised to the one line law, especially in flat wooded country:

(1) The difficulty of finding such a line.

(2) The difficulty of determining, from the posts, the area covered by the staked claim.

(3) The possibility of prospectors unknowingly staking the same area.

(4) The ease with which country may be blanketed.

The first point mentioned needs no elaboration. Having found the location line it is usually impossible, after a few months, to decipher the writing giving particulars of the distances to right or left of the location line to which the claim extends. The claim maps of the recording office are almost useless, for claims may be actually miles from the position shown on these compilations made from prospectors' sketches.

In dense bush it is possible for prospecting parties to stake practically the same area of ground without either one knowing of the staking by the other. And, finally, during the boom days of a new camp the one line method lends itself to easy and quick blanketing of a country without the stakers even being forced to leave the canoe routes.

Objections may be raised that the four line staking of the Ontario law requires more time and labour than does the one line location of the Dominion law. Surely if a claim is worth staking at all it is worth cutting a line around the four sides of it. And from the viewpoint of those who are not fortunate enough to be first in an area there is no question about which plan is the more satisfactory. Uncertainty as to whether or not an area is open often leads prospectors to abandon promising areas for virgin fields. The area of pre-Cambrian rocks in which valuable minerals are likely to occur in Manitoba is much smaller than that in Ontario. It seems unfortunate that a further handicap should be placed on the western province by a method of staking that is not adapted to give every real prospector a fair chance to locate and develop his claim in security.

# THE MINERAL INDUSTRY OF BRITISH COLUMBIA

By Stuart J. Schofield.

Annual Western Meeting, Vancouver, November, 1922.

The mineral production of British Columbia, as given by Mr. W. Fleet Robertson in the Annual Reports of the Minister of Mines of British Columbia, is shown in the table on the following page.

An examination of this table for the years 1900 to 1919 inclusive will show that the average yearly production of copper in pounds has been 41,766,498 and a comparison of this and the yearly output will show that the production of copper has not increased very appreciably during the last fifteen years. The same is true of the lead, silver and lode gold production which, for the same period, has averaged 42,836,983 lb., 3,286,352 oz., and 219,919 oz., respectively. The most outstanding increase in production is credited to zinc, which increased from 4,184,192 lb. in 1910, the first recorded production, to 56,737,651 lb., in 1919. This increase was not due to the discovery of a new zinc deposit but to the discovery of a successful metallurgical treatment of the ore from the Sullivan mine, discovered in 1896.

That the production of the basic metals, copper, lead, silver, and gold, in a young and growing country should remain approximately stationary is remarkable. Another significant fact is the steady decline in prospecting. This decline has been realized both by the Provincial and Dominion Governments, and the former has taken measures to remedy the situation, but so far without any tangible results.

The mining industry is one of the basic industries of British Columbia. Its growth has far-reaching effects on the development of the province as a whole, especially in regard to transportation. It has been stated by no less an authority than the present Deputy Minister of Mines for the Dominion

| Year | Copper (pounds) | Lead (pounds) | Zinc (pounds) | Silver (oz.) | Gold oz. Lode | Gold oz. Placer |
|---|---|---|---|---|---|---|
| 1900 | 9,997,080 | 63,358,621 | | 3,958,175 | 167,153 | 63,936 |
| 1901 | 27,603,746 | 51,582,906 | | 5,151,333 | 210,384 | 48,505 |
| 1902 | 29,636,057 | 22,536,381 | | 3,917,917 | 236,491 | 53,657 |
| 1903 | 34,359,921 | 18,089,283 | | 2,996,204 | 232,831 | 53,021 |
| 1904 | 35,710,128 | 36,646,244 | | 3,222,481 | 222,042 | 44,765 |
| 1905 | 37,692,251 | 56,580,703 | | 3,439,417 | 238,660 | 48,465 |
| 1906 | 42,990,488 | 52,408,217 | | 2,990,262 | 224,027 | |
| 1907 | 40,832,720 | 47,738,703 | | 2,745,448 | 196,179 | 41,400 |
| 1908 | 47,274,614 | 43,195,739 | | 2,631,389 | 255,582 | |
| 1909 | 45,597,245 | 44,396,346 | | 2,532,742 | 238,224 | |
| 1910 | 38,243,934 | 34,658,746 | 4,184,192 | 2,450,241 | 267,701 | |
| 1911 | 36,927,656 | 26,872,397 | 2,634,544 | 1,892,364 | 228,617 | |
| 1912 | 51,456,537 | 44,871,454 | 5,358,280 | 3,132,108 | 257,496 | |
| 1913 | 46,460,305 | 55,364,677 | 6,758,768 | 3,465,856 | 272,254 | |
| 1914 | 45,009,699 | 50,625,048 | 7,866,467 | 3,602,180 | 247,170 | |
| 1915 | 56,918,405 | 46,503,590 | 12,982,440 | 3,366,506 | 250,021 | |
| 1916 | 65,379,364 | 48,727,516 | 37,168,980 | 3,301,923 | 221,932 | 29,025 |
| 1917 | 59,007,565 | 37,307,465 | 41,848,513 | 2,929,216 | 114,523 | 24,800 |
| 1918 | 41,772,916 | 43,899,661 | 41,772,916 | 3,498,172 | 164,674 | 16,000 |
| 1919 | 42,459,339 | 29,475,968 | 56,737,651 | 3,403,119 | 152,426 | 14,325 |
| 1920 | 44,887,676 | 39,331,218 | 49,208,268 | 3,377,849 | 120,048 | 11,080 |
| 1921 | 41,402,288 | 41,402,289 | 49,419,372 | 2,673,389 | 135,663 | 11,660 |
| 1922* | 32,400,000 | 67,500,000 | 57,100,000 | 7,100,000 | 198,000 | $365,000 |

*Estimated

that over 50% of the freight hauled by the railways is to be credited to this industry.

It is the purpose of this paper to analyze the various factors which have a bearing on the development of the mining industry in British Columbia, and to suggest measures whereby its healthy growth may be stimulated.

Minerals, like every commodity which enters the world's markets, are directly affected by competition with the various countries producing them. The ability to produce at low cost is the essential element in this competition. The keenest competitors of British Columbia for capital to be used in the mining industry are Mexico and the western United States.

According to the Mexican Year Book for 1920-21, the production of gold and silver in that country increased from 21,846,000 in 1890 to $65,181,000 in 1910. Copper, in the same twenty-year period, increased from 18,000 tons to 60,000 tons, and lead and zinc in proportion.

The metal production from the State of Utah, a typical desert state, with typical desert ores, is as follows:

| | Gold (lode) (oz.) | Silver (oz.) | Copper (lb.) | Lead (lb.) | Zinc (lb.) |
|---|---|---|---|---|---|
| 1865-1880 | 150,837 | 33,873,227 | 3,754,194 | 350,713,000 | |
| 1881-1890 | 129,508 | 56,137,249 | 11,830,773 | 556,316,000 | |
| 1891-1900 | 858,135 | 75,346,301 | 47,349,628 | 743,792,000 | |
| 1901-1910 | 2,092,325 | 111,068,934 | 624,858,979 | 1,130,541,875 | 2,637,655 |
| 1911-1917 | * 1,273,841 | 89,594,816 | 1,272,368,839 | 1,194,237,282 | 13,212,239 |

| | | |
|---|---|---|
| Value | 1865-1880 | $ 64,155,452 |
| | 1881-1890 | 87,101,029 |
| | 1891-1900 | 106,606,912 |
| | 1901-1910 | 252,820,473 |
| | 1911-1917* | 405,471,471 |

The rapid growth of the mining industry in Mexico and in such states as (say) Utah, to select an example at random, in

*Seven years

comparison with the slow growth of this industry in British Columbia, shows us clearly that it is difficult for British Columbia to compete for capital for the mining industry.

In order to understand the situation thus presented, it will be necessary to examine carefully the various factors which have a bearing on the mining industry. These may be considered under the headings (1) Prospecting, and (2) Development and Production.

*Prospecting.* — The topography of British Columbia is so well known to us locally that we realize the inaccessibility of a large part of our province. The mountains are high and rugged and, for the most part, are covered with a dense vegetation in the lower and more accessible portions. In addition, a large part of the province is covered with snow for six months of the year. The latter factor reduces by one half the time for prospecting, and the covering of vegetation is ever with us, making travelling very difficult and the construction and maintenance of trails and roads very expensive. On the other hand, the numerous and deep fiords which cut into the Coast Range and sometimes penetrate even into the interior of British Columbia, afford unrivalled avenues for prospecting and transportation. The numerous lakes and rivers which are so abundant in British Columbia are of inestimable value to the prospector, as may be seen from the fact that mining activity has, in the past, centred largely around such avenues as Kootenay, Slocan and Moyie lakes. The more recent rapid development of mining in southern British Columbia was largely the cause of the building of the Crowsnest and Kettle Valley railways. The close relation between transportation and mining activity is apparent to all. The thick layer of drift that mantles the valley slopes and covers the valley floors effectively prevents the discovery of mineral deposits. Most of the discoveries of mineral deposits in British Columbia have been made at high elevations, where the rocky peaks and ridges are free from drift and vegetation.

On the other hand, the desert regions of the western United States and Mexico are characterized by a more or less rolling topography and are almost devoid of vegetation. Hence prospecting is comparatively easy in spite of the heat

in the summer time, and the absence of water which, however, is easily carried in tanks on wagons. The desert countries have not been glaciated to any extent and hence the mantle of glacial drift, which is such a handicap to prospecting in British Columbia, is absent, so that the mineral deposits are readily found even on the lower slopes of the valleys.

In summing up, it can be readily seen that prospecting in British Columbia is an arduous undertaking, surrounded by many natural disadvantages; conversely, in the desert regions of Mexico and the western United States, it is pursued very quickly and rather comfortably throughout all seasons of the year. We should not lose sight of the fact, however, that, during the last few years, very few, if any, discoveries of new mining districts have been made in these desert regions, any discoveries being within well-known districts and resulting from costly mining exploration. Apparently these regions have been pretty thoroughly prospected. But British Columbia, as well as the greater part of Canada, is almost unprospected on account of the natural difficulties as above enumerated. It is safe to predict that, not only will new deposits be discovered in the present known mineral districts, but that new mineral districts will be found as transportation develops and the reserves in the deposits of desert regions become small and are finally exhausted. Mining men do not expect that many new mining districts will be found in the desert regions of the western United States and Mexico. If found at all, they will be in regions that will rival British Columbia in their natural disadvantages to prospecting.

This confidently expected increased demand for metals from British Columbia will increase the sales of mineral claims, which, in turn, will immediately react by creating an increased interest in prospecting. Prospecting can be stimulated in a natural way only by an increased demand for mineral claims. We cannot make it too clear that the law of supply and demand as applicable in the case of mineral claims as with other products.

*Development and Production.* — Once a mineral outcrop is located, the process of development to determine its possibilities commences.

The mineral deposits of British Columbia are sharply contrasted, in their mineralogical composition, with those of the desert regions. The British Columbia deposits are primary sulphides; those of the desert regions being the oxidized and enriched products of primary sulphides.

All ore deposits of the metal type at the time of their formation are called primary; that is, the minerals of these ore-deposits have been separated through the agency of mineralizing solutions from the parent magma for the first time. Usually the metals are in the form of sulphides, like chalcopyrite, galena, zinc blende etc.; sometimes as oxides, such as magnetite, chromite, etc.; and in the case of gold, as the metal itself. Usually the primary sulphide ores are not rich enough in the several valuable metals to permit of their being shipped, as mined, to the smelter. They must be concentrated by separating out not only a large quantity of the gangue material, but also the valueless sulphides such as pyrrhotite and pyrite. It is to this type of deposit that most of the ores of British Columbia belong. Turning now to the desert regions of the western United States, the separation and a partial concentration have been effected by the action of descending waters on the primary sulphides. Thus these deposits are not only richer in metal content but, also, the pyrite and pyrrhotite have oxidized to form oxides of iron — a decided advantage in the subsequent process of smelting, especially of lead ores. The problem of selective flotation is almost completely eliminated. But the main factor is the increased metal content of the enriched secondary sulphide zones, which, in the end, greatly reduces the cost of production. In the case of the enriched copper ores, crushing and concentration are simple matters accomplished at small cost. The oxidized silver-lead ores need no concentration, and are shipped direct to the smelter.

The presence of deep oxidized, and oxidized and enriched zones is related to the depth of the ground water — that blanket of water which occurs at various depths below the surface of the ground. In British Columbia, on account of the heavy precipitation, the ground water is usually close to the surface, so that the primary sulphides occur practically at the outcrop of a mineral deposit. In the desert countries,

however, on account of the slight precipitation, the ground water level is sometimes 2,000 feet or more below the surface, so that a deep oxidized zone, with or without enrichment, is present.

It is evident, from the occurrence of the enriched silver deposits of the Premier and Dolly Varden mines, that oxidized and enriched ore bodies existed in British Columbia before the Glacial period, whose effects, in North America, were restricted almost entirely to Canada, including British Columbia. Glaciation, however, has largely removed these oxidized deposits, and has left only a small portion of the enriched zone, as in the case of the Premier and Dolly Varden mines, which, from our present knowledge, appear to be exceptions to the general character of British Columbian ore deposits, composed, as they usually are, of primary ores.

In the desert regions, glaciation was limited to the highest altitudes, and even here it was not intense, so that both the oxidized and enriched portions of the ore deposits have been preserved.

These valuable oxidized and enriched portions, which can be mined at the very beginning of operations, place the mine upon a paying basis very early in its history. Later, when the zone of primary sulphides is reached, the mine is in a condition, if properly managed, to handle the low-grade sulphide ores, which is a more expensive process. Because of the ruggedness of the topography, the heavy vegetation, and the deep snow in winter, in British Columbia, the construction of roads, railways, tram lines, and power lines so necessary in the development of a large mine is much more costly here than in the desert regions.

On the other hand, the heavy precipitation and snow-fall in British Columbia furnish the cheap power so essential to mining operations.

Thus, it can be readily seen that the development of the mining industry of British Columbia labours under certain natural disadvantages, which have been reflected in its relatively slow growth.

Naturally, large mining companies prefer those properties where no great outlay of capital is required, and no great lapse of time occurs, before production can be secured. This can be readily understood, for the changing conditions of the markets make it impossible to predict with any degree of certainty what the price of metals will be one or two years in advance. In the desert regions, production is secured at an early stage in development, as pointed out above, while in British Columbia, production is secured only after a large outlay of capital and after a large amount of development work has been accomplished.

In the case of Mexico, unsettled conditions of the country and the restrictive legislation of a labour government in part offset the natural advantages of this large territory.

In the desert regions of the United States, the mineral industry has very little difficulty in securing capital for its development. Great natural advantages, good labour conditions, and a steady government, place this territory in a peculiarly favourable position from the investor's standpoint.

Having pointed out certain handicaps under which the mining industry labours in British Columbia, it remains to consider in what way these handicaps can be overcome.

Natural conditions cannot be changed. Hence it is absolutely necessary that mining in British Columbia be carried on with strict economy and under the most rigid scientific control. Scientific research in geology and metallurgy, to a degree undreamed of in this country at the present time, must be undertaken at the earliest possible moment.

The Federal Department of Mines, including the Mines Branch and the Geological Survey, has carried on some research work in connection with the ore deposits of British Columbia. But the country is so large and the problem so complex that this small band of earnest workers cannot begin to complete the investigations of the mineral areas already discovered, let alone do much exploratory or reconnaissance work in new and unexplored areas. At the present time there is great need for an expansion of the work of the Geological Survey and the Mines Branch. The crying need to-day is for maps, both

geological and topographical, that can be used by the prospectors and other pioneers who are largely responsible for the development of new territory. If prospectors were able to have the topographical and geological maps before entering an undeveloped area, it would greatly facilitate their work.

The mining companies of British Columbia, with the exception of Granby and Britannia, have not, as yet, employed the aid of geology in the exploration and development of their mining properties.

In a country such as British Columbia, where the location and study of ore deposits is a difficult and complex problem and where ore deposits peculiarly lend themselves to the aid of such research, geology should be an essential adjunct to mining. On the other hand, the large mining companies of the United States have highly organized geological staffs which carefully study the underground geology as development work proceeds. The high degree of efficiency to which these departments have been developed is a tribute to the foresight and business acumen of these enterprising mining companies. At the Annual Western Meeting of the Institute in Vancouver in 1920, Mr. Billingsley, of the Anaconda Copper Mining Company, made the following statement:[1] "In the present state of our knowledge of the Butte structure, it is almost invariably true that a vein is found as predicted on geological grounds. The finding of ore in the vein used to be a fifty-fifty bet. In recent years, however, as we have come to pay more attention to the position and pitch of ore shoots within the veins, commercial ore is found, I think, in about 85% of the work recommended."

The discovery in 1910 of oil flotation as a concentrating process for sulphide ores revolutionized mining in British Columbia to an extent hardly dreamed of by those who have not studied the history of mining in the province.

Britannia, staked in 1898, did not become a producing mine in a commercial way until flotation was employed. The ordinary wet methods of concentration had proved failures. Copper Mountain, discovered in 1896, remained undeveloped until the concentration of this ore had yielded to flotation.

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(1) Trans. Can. Min. Inst., Vol. XXIII, 1920, p. 362.

Therefore every endeavour should be made to encourage research on ore concentration, in order to discover new methods and to improve methods already in use. The Sullivan mine, started in 1895, was unsuccessfully operated from 1902 to 1908 because of the complexity of the ore. Utilization of the great quantities of zinc ore in this mine was made possible only by intensive research on these ores. Previous to 1910 no zinc was produced in British Columbia. The treatment of the Sullivan ore by differential flotation, and the electrolytic zinc process, brought to a high state of efficiency by the metallurgists of the Consolidated Mining and Smelting Company, have made possible an annual production of approximately 50,000,000 pounds. Scientific investigation again proved its value in a commercial way.

There is still another field of investigation which at the present time remains practically untouched; that is, the location of ore bodies by methods which the modern development of physics has brought to light. The outlines of bodies of magnetite below the surface have been drawn with a fair degree of accuracy by the magnetometer, and it is by the investigation of such phenomena as magnetism and radio-activity that more accurate methods of location and outlining of ore bodies may be discovered.

This paper is presented merely to open up the discussion on the mining industry of British Columbia, and it is hoped that it will be followed by other papers which will deal with other phases of this problem, a solution of which is so important economically to British Columbia.

# QUARTZ VEINS OF BARKERVILLE, CARIBOO DISTRICT, B.C.[1]

By W. L. Uglow

Annual Western Meeting, Vancouver, November, 1922.

## Introduction.

Ever since the commencement of placer mining in the Barkerville area, in the early sixties, the extreme richness of the Tertiary stream gravels, and the coarseness, angularity and crystal forms of the gold which has been recovered from these gravels, have given rise to much speculation as to the source of the precious metal. It was recognized by the early miners, and emphasized by Bowman[2] in 1887, that the streams which carried the richest coarse gold had their sources in Bald Mountain plateau and its adjoining spurs, and that the richest portions of those streams were immediately adjacent to this plateau.

After the exhaustion of the richest ground in the seventies, the attention of the miners was directed towards the discovery of the mother lode of the gold. Bald Mountain plateau revealed the presence of a belt of large quartz lenses and veins, richly mineralized in places with pyrite, and considerable exploration was devoted to these veins. Failure to discover bonanzas which might have fed the placers gradually led to the abandonment of the search.

The constantly diminishing production of gold from the gravels during the last few decades has brought a realization of the fact that the placer possibilities of the Barkerville area are limited, and are at the present time largely confined to dredging ground and a few areas of hydraulicing ground. The future life of the camp as a continuous mining community is inevitably bound up with the possibilities of the quartz veins.

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[1] Published by permission of the Director, Geological Survey of Canada.

[2] Ann. Rep., Geol Surv Can., 1887.

The following discussion is a summary statement of these possibilities based on a detailed examination of the area made by the writer for the Geological Survey of Canada during the season of 1922.

## LOCATION AND TOPOGRAPHY OF THE AREA.

The area from which the wealth of Antler, Williams, and Lightning creeks, and their tributaries, has been derived, is very limited in extent, and most of it may be enclosed in a rectangle measuring ten by fifteen miles. Barkerville and Van Winkle (Stanley), the two chief communities of the district, are located 58 and 44 miles, respectively, east of Quesnel, the present terminus of the Pacific Great Eastern railway, on the bank of the Fraser river. They are connected with Quesnel by means of a motor road.

The topography is that of a highly dissected upland plateau, the elevations of the inter-stream ridges ranging from 5,500 feet to 6,500 feet, while the valley bottoms average around 4,000 feet. With few exceptions the ridges are well timbered and the rocks are obscured by a thin mantle of glacial drift.

## GENERAL GEOLOGY.

The bedrock geology consists chiefly of two series of metamorphosed sedimentary rocks — the Cariboo series and the Bear River series, the latter of which is the younger—and two intrusive formations, to be known as the Mount Murray diabase sills and the Proserpine quartz porphyry.

The sedimentary series are broadly folded into an anticlinorium which seems to pitch to the southeast. The average strike of the rocks is N.55°W. The main axis of the fold runs through Bald mountain and Mt. Agnes, and crosses Lightning creek near its source. The Cariboo or gold-bearing series consists of massive to thinly-bedded quartzite or quartz slate, sericite schist, black slate and fine-grained blue-grey limestone. The quartzites are micaceous and are loosely called schists. The proportion of true schists in the Cariboo series of this area is small. The Bear River series consists of conglomerate, crinoidal limestone, argillite, and thinly bedded chert.

The Proserpine quartz porphyry occurs as sills and dikes of comparatively small size, lacks schistosity, and is in most places much iron-stained. It was found cutting the Cariboo series only, and is intimately associated with the quartz veins.

The Mount Murray diabase occurs chiefly as sills in the argillites and cherts of the Bear River series. It is both coarse and fine-grained, and is generally quite unaltered. In a few places a similar rock was found as small dikes intruded into the Cariboo series.

The quartz veins of the district are areally related to the Cariboo series. They occur in the quartzite members chiefly, especially in the more thinly-bedded, schistose portions. The veins appear to be absent from the Bear River series. This field fact makes the distinction of the two series a matter of great importance from the standpoint of any future mining exploration.

## HISTORY OF THE QUARTZ MINING.

Commencing with 1877 and 1878, the prospectors turned their attention to the exploration and development of many of the quartz veins. The Provincial Government established a reduction works at Barkerville for the treatment of the quartz ores, and many shafts and tunnels were driven. Considerable free gold was derived from the oxidized portions of the veins, but very little recovery was made from the sulphides. None of the deposits was scientifically explored or exhaustively tested. Interest in the vein deposits gradually "petered out" in the early nineties, and was only re-awakened in 1919 when the Mining Corporation of Canada bonded several claims on Proserpine mountain, and spent many thousands of dollars on surface exploration during the following winter. Without having carefully tested the mountain, and, manifestly, without having recognized the importance of the intersections of the vein systems, they relinquished the bond in the spring of 1920. Since that time only minor amounts of work have been done on the veins. Detailed accounts of the progress of the quartz mining may be found in the earlier reports of the British Columbia Minister of Mines, and summary statements of the character of veins are available in Amos Bowman's Report

on the Geology of the Mining District of Cariboo (Geological Survey of Canada, Annual Report for 1887)

## CHARACTER OF THE VEINS.

A fact of considerable significance in any estimated or prophecy regarding the quartz possibilities of the Barkerville district is that there are at least two very district types of veins. These have been called by the writer the "A" and the "B" veins.

*The "A" Veins.*— Tnis set of veins constitutes one of the most conspicious features of the camp. They are usually large bodies of white quartz varying in width up to 100 feet, and of unknown longitudinal extent. They are especially prominent in a broad zone of shearing which strikes about N.35°-40°W. nearly parallel to the trend of the formations. This zone varies in width from one quarter of a mile to one mile or even more. Longitudinally it has been traced from Round Top mountain on the southeast to Sugar Creek on the north-west — a distance of over 25 miles. This quartz outcrops conspicuously on the grassy uplands of the plateau surface — in places standing as much as ten feet in relief. It seems probable that individual bodies of quartz in this zone are quite discontinuous and lenticular, but the country rock in their vicinity is usually well charged with small bunches and stringers of quartz of the same character.

These veins are abundantly intersected by crossing closely-spaced fractures, so that the quartz is exceedingly friable and tends to break into plate-like pieces. This fact and the "*en échelon*" or step-like arrangement of adjoining lenses indicates that these veins are of pre-schist origin; that is to say, after their formation, they were involved in the stresses which folded the rocks and formed the shear zone.

Veins of this class are only sparsely mineralized with sulphides. In places, they appear to be quite barren, while in other places they are impregnated with pyrite, which usually is found in strongly cubical development. Other sulphides are typically absent. The country rock in the vicinity of the pyritized portions of the veins is usually richly studded with pyrite crystals.

No primary or originally free gold was found to occur in these veins. The only assays available at the time of writing indicate that the pyrite carries only a low content of gold.

The following may be cited as examples of this class of veins: (1) The Bonanza vein, of the B.C. Milling and Mining Co., near the head of Stout's gulch. Along its outcrop, this vein strikes N.45°W., and consists of 18 feet of banded white to pinkish quartz containing very small amounts of pyrite. Material now lying on the dump proves that, in depth, portions of the quartz are thickly studded with pyrite, in crystals and bunches. The vein appears to be a strong ledge, but, in the distance for which it was followed for mining, no shoots of high-grade sulphides were encountered.

(2) The large "blow out" of quartz exposed in the hydraulic pit in Stout's gulch, which is now exposed for a width of 50 feet, and a length of 150 feet.

(3) Quartz ledges on the top of Mount Burdett, 7 miles south of Barkerville. Some of these are continuous for one-quarter to one-half mile, and vary up to 100 feet in width. Only very small amounts of pyrite can be seen in these veins.

(4) Quartz ledges which outcrop on the top of Bald mountain, of the same general character as the last.

Very little accurate information is at present available concerning the values in the veins of the "A" class. It is well recognized by those who have mined the quartz that no free gold has been found or is to be expected after the bottom of the zone of oxidation has been reached. Whatever gold the veins contain is inherently associated with the pyrite. Only low values can be expected from them, but large tonnages of quartz could easily be mined. Small amounts of finely crystalline gold may be washed from the oxidized portions of the veins, which indicates that the sulphides are at least somewhat auriferous.

*The "B" Veins.* — The "B" veins are inconspicuous features of the district, yet they appear to be the ones with which the precious metal values are predominantly associated and to which any mining developments that may materialize in the future will be entirely due.

The rocks of the Cariboo series are strongly cross-jointed in a direction varying between N. 25°E and N. 35°E. Some of these joints are faults of appreciable displacement, while others are simply fractures so closely spaced as to give the rocks a marked cleavage in those directions. This system of fracturing crosses the "A" veins and is the means of producing in them the sheeted structure already mentioned.

These northeasterly-trending fractures are usually healed with quartz, perhaps only of the thickness of a sheet of paper. Quite commonly, however, the quartz fillings are 12 to 15 inches wide, varying up to 2 feet. In many places series of multiple veins occur, consisting of a number of parallel veins close together.

The quartz of the "B" veins is easily distinguishable from that of the "A" veins, by the fact that it is massive and unsheared. The marginal portions of these veins are characterized by narrow comb-like bands of siderite and ankerite which are weathered to a yellow or dark brown colour near the surface.

The veins are commonly well mineralized with galena, pyrite, and arsenopyrite, with minor amounts of sphalerite, barite, and scheelite. The sulphides carry high gold values in many cases. The un-oxidized portions of the veins exhibit cavernous honeycombed quartz from which pyrite crystals have been leached, and in many places coarse crystal gold has been found in these cavities.

The following cases may be mentioned as examples of the "B" type of veins:

(1) The series of northeasterly-trending veins on the Proserpine surveyed claim, near its southwestern line, on one of which the old Forrest shaft was sunk to a depth of 60 feet. The veins vary in width up to 2 feet, and are well mineralized with pyrite, arsenopyrite, and galena.

(2) The series of northeasterly-trending veins on the Warspite and Tipperary surveyed claims on Proserpine mountain. These are exposed only by surface trenching, and consist of quartz, well mineralized with galena, arsenopyrite, and pyrite.

(3) The Beede or Perkins ledge on Burns mountain, about two miles east of Stanley (Van Winkle). Here there are several parallel northeasterly-trending veins, varying in width up to 2 feet, containing pyrite and galena, with considerable free gold in the upper oxidized portions. The old workings, consisting of shafts and drifts, on these veins are now full of water and inaccessible.

(4) The scheelite-bearing veins of Hardscrabble creek,[1] 7 miles northwest of Barkerville. These are quartz-ankerite veins carrying pyrite, galena, and scheelite. The workings are not accessible at the present time.

*Intersections of the "A" and "B" Veins.*—Veins of the "B" type are post-schist in origin. They cross-cut the foliation of the schistose quartzites and are not involved in any shearing. They are found meeting and cutting through the "A" veins. From a study of several of these intersections it is inferred that mineralizing solutions travelling along the "B" set of joint fissures not only mineralized the fissures themselves but impregnated the veins of the "A" type near their intersections with the group of sulphides above-mentioned. A sufficient number of these intersections was seen to establish the fact that definite shoots of ore of fair size are to be expected at these points. Samples taken by the writer from these shoots have not yet been assayed (at the time of writing), so that it is impossible to give any figures for values.

The following cases will serve as examples of the kind of shoots to be looked for at these intersections:

*Black-Jack.*— This property is located on the southeast bank of Williams creek at the mouth of Black Jack canyon, at the south end of the town of Barkerville. The belt of "A" veins is here intersected by a large number of mineralized cross fissures of the "B" type, which have formed a well defined shoot. Twenty-five mineralized cross stringers from 2 inches to 10 inches in thickness were counted in a total width of 50 feet. A shaft, 120 feet deep, is on the property, but it is now caved and full of water. The records of the old Black Jack

(1) Report on Tungsten Ores of Canada, by T. L. Walker, Mines Branch, Dept. of Mines, Canada, Pub. No. 25, 1909.

Mining Company show that from 202 tons of material taken from the 42-foot level, $3,573.10 in gold was recovered — an average of $17.71 per ton. Of this amount $2,646.10 was recovered from 46 tons of sulphide concentrates at the rate of $57.52 per ton. The property belongs to F. J. Tregillus and T. A. Blair of Barkerville.

*Independence-Kitchener.*— These are two adjoining claims situated on Proserpine mountain at elevations of 5,600 to 5,700 feet. The Independence is recorded in the name of E. E. Armstrong, and the Kitchener in the names of F. J. Tregillus, T. A. Blair, and P. Carey. Both are surveyed claims.

Both claims are located on part of the main zone of "A" veins, where quartz lenses of the "A" type are intersected by the narrower quartz-ankerite-siderite veins of the "B" type. In the vicinity of the intersection, the "A" quartz is heavily mineralized with arsenopyrite, foliated galena, and pyrite. A tunnel of shallow depth and three small open-cuts are the only workings on this intersection. Insufficient work has been done to show the extent of the shoot of sulphides, either horizontally or vertically, but the intersection of the vein systems is quite clear, and a high percentage of galena and arsenopyrite can be seen in the faces of the open-cuts and in the several hundred pounds of vein material lying on the dumps.

*Victory.*— This property, recorded in the name of A. W. Sanders, is also located on Proserpine mountain. Several small open-cuts reveal veins of the "A" and "B" types. Barren-looking white shattered quartz of the "A" class is exposed in two open-cuts for a width of 11 feet. These exposures belong to the main belt of "A" vein mineralization. About 500 feet to the north of these cuts are two others which have been driven on veins of the "B" type. These veins are individually narrow, but they are closely spaced, and are exceptionally well mineralized with galena, arsenopyrite, and some sphalerite. The intersection of the two vein systems on this claim has not been uncovered, but there is every likelihood that a shoot of sulphides would be found to occur there.

*Continuation of Values with Increasing Depth.*— There is a popular conception quite current among the placer miners, and even amongst others who are primarly interested in the lode deposits, that the reason why the coarse nuggety gold of the gravels has only to an exceedingly limited extent been encountered in the veins, is that the veins have undergone such deep secular decay and erosion that the upper primarily rich portions of the lodes were removed in former geological periods and that only the roots remain. This hypothesis has an element of truth in it, but it is founded on a misconception, as will appear from a subsequent section.

A diagnosis of the mineralogy and structure of the veins proves that there is no apparent reason for any rapid change in the character of the minerals or in their gold content within mineable depths. The presence of scheelite and minor amounts of pyrrhotite and tourmaline in the veins of the "B" type strongly suggests that the solutions which deposited the minerals were of intermediate to high temperature, and there is no indication of near-surface conditions of deposition. The solutions certainly owe their origin to deep-seated intrusive rocks, evidence of whose existence is given by the occurrence of numerous quartz porphyry and aplite dikes and sills. The veins are found to be genetically associated with these minor intrusions.

The misconception mentioned above is based on the failure to differentiate between the primary or original gold content of the veins as present in the sulphides, and the free visible gold of the upper portions of the veins, which is a secondary enrichment due to the oxidation of the sulphides.

*Relationship of the Free Crystal Gold of the Upper Parts of the Quartz Veins to the Auriferous Sulphides at Depth and to the Gold of the Placers.*— Fine- to coarse-grained crystalline gold, frequently in the form of definite single crystals, groups of crystals, wires and plates, has been recovered by washing the upper oxidized portions of many of the quartz veins of the district. Two masses of this type of gold, said to have been worth $120 and $80, were obtained from the veins of Burns mountain about the year 1880. To-day the chief method

of locating auriferous lodes is by panning the disintegrated and decomposed country rock and vein matter. In many cases the amount of gold so obtained is very small and is in the form of exceedingly fine colours, which are probably crystals. In other cases, very encouraging amounts of coarse ragged gold, associated with small grains of quartz, are recovered in the pan.

Several times during the past season, the writer had occasion to visit the Rainbow prospect on the east slope of Cow mountain, where Mr. A. W. Sanders was opening up a prospect. This prospect consists of the intersection of a few shattered and disintegrated quartz stringers in decomposed and broken schistose quartzite. There are two small open-cuts on the property, and from these several panfuls of broken and rotten rock were taken in the presence of the writer. The material was rapidly and roughly panned without being crushed. The gold in two pans was weighed and found to have a value of $1.57. Other pans were estimated to contain upwards of $2.00 each.

From the vein material below the zone of oxidation, where it has been encountered in the old shafts and in one new one on the Warspite claim, no such free gold was recovered. The reason is, doubtless, that the sparsely disseminated gold which was originally present in the iron sulphides of the unaltered vein material was released by the oxidation of the sulphides and appeared as free gold crystals in the upper disintegrated parts of the veins. By an alternating process of solution and re-deposition, the gold grew into fairly large-sized grains, crystals, wires and plates, while the pyrite and other iron sulphides were preferentially carried away in solution. Accumulations of gold of this character in the upper parts of the veins supplied the material for the gravels.

Without, on this occasion, going into the reasons for the statement, it may be said that the weathering of the veins which supplied the material for the nuggets was of Tertiary age, and a large part of the free gold still found in the upper portions of the veins occurs at the roots of the Tertiary zone of oxidation, which escaped removal by the continental ice sheet.

## POSSIBILITIES OF THE QUARTZ VEINS.

The facts which control the possibility of successful quartz mining in the district are the following:

(1) The large irregular lodes of the "A" type will not contain, throughout their extent and breadth, sufficient gold values to make the mining of them profitable.

(2) The highly mineralized veins of the "B" class are as a rule too narrow to be mined alone, as the tonnage of ore that could be taken from them would be small.

(3) The unproved possibility which the district possesses is connected with the intersections of the two vein systems. It is possible that a careful and systematic exploration of some of the "A" veins, with especial reference to zones of intersecting "B" veins, might reveal shoots of auriferous sulphides sufficiently closely spaced and of satisfactory grade for commercial mining.

(4) The failure to recognize the importance of these intersections may have been largely responsible for previous failures to develop ore bodies.

(5) The future of any quartz mining enterprise will have to depend entirely on the auriferous content of the sulphides, and not on the free gold present in the upper parts of the veins.

(6) Any future exploration should proceed by the methods of locating the intersections and sinking on them. Surface work of less than 30 or 40 feet in depth is valueless, except for proving the horizontal extent of the veins.

## DISCUSSION.

MR. M. F. BANCROFT: Most of the facts enumerated by Professor Uglow can be tested by a prospector of average intelligence, as they are based on first-hand field observations. The paper deals with two fissure systems, containing quartz veins easily distinguishable from the standpoint of structure and showing marked differences of mineralization. The problem of ore deposition in these veins is bound up with regional problems of deformation and igneous intrusion.

I am particularly interested in what Dr. Uglow says in respect to the mineralization of the "A" veins, which form a conspicuous feature in the Barkerville area. The longitudinal type of quartz vein is abundantly evident in the upturned strata of the Lardeau area, and in the same formations north of the main line of the Canadian Pacific Railway. In their strike, these quartz veins approximate the trend of the formations, and they are commonly referred to as "formational leads". Many of them show only a sparse mineralization such as that described by Dr. Uglow for the "A" veins of Barkerville, but in the Selkirk region longitudinal veins give rise to excellent surface showings of lead and zinc sulphides, quite apart from the conditions of vein intersection.

The point I wish to make clear is that, outside the Barkerville area, well mineralized bands or veins were formed at different stages in the progressive deformation of the rocks. The "A" veins belonging to the Cariboo series may have formed at a period of little mineralization, and have then been sealed up and shifted about with the folding and shearing of the host rock. The source of these veins is not clear in their present relation.

The situation is somewhat more definite in the Lardeau area. Longitudinal veins and silicified mineralized bands occur in this region associated with altered and sheared minor igneous intrusives. Both quartz veins and dykes have been subjected to intense dynamic metamorphism. The quartz of these veins is commonly found to be sheared, shattered, and granulated.

The quartz veins in the central mineral belt of the Lardeau area are roughly classified as gold-quartz veins and galena veins, but it does not appear to be a very significant distinction. Good gold values are obtained from some of the galena ores. There would seem to be a close genetic relation between the quartz veins and the altered intrusives of the central mineral belt.

The rocks of the central mineral belt in the Lardeau contain abundant evidence of igneous activity throughout the latter part of the Carboniferous. It is quite probable that a fair amount of ore deposition took place at this early stage in the geological history of the region.

Attention has been focussed on events in the Mesozoic that led to ore deposition in the Kootenay district. The possibility of pre-Cambrian mineralization being an important factor has lost force in recent years, and little thought has been given to the possibility of pre-Mesozoic mineralization.

Professor Uglow states that the "A" veins are of pre-schist origin, and, in the sense in which he uses the term pre-schist, the meaning is clear. However, the term pre-schist may be used a little more guardedly in view of the fact that *schist-producing* stresses may have operated on the Cariboo series prior to the formation of the "A" veins, though it might be a difficult matter to prove that such was the case.

In view of the facts stated by Dr. Uglow concerning free gold in the quartz veins of Barkerville, no additional light seems to have been thrown on the origin of the placers. The concentrations of gold in river gravels can still be safely considered to be a feature of weathering, the free gold, fine or coarse, being left in the region as a residual product of erosion.

# GOLD-DREDGING POSSIBILITIES IN THE BARKERVILLE AREA, BRITISH COLUMBIA[1]

By W. A. Johnston.

Annual General Meeting, Ottawa, March, 1922.

## Introduction.

Placer mining has been carried on to some extent in the Cariboo district, British Columbia, almost continuously since the discovery, over 60 years ago, of the gold-bearing creeks—Antler, Williams, Lightning, and several others — which have rendered the district famous. The main reason why these placers have had an exceptionally long life, as compared with many other placer mining camps, is that the area was glaciated during Pleistocene time; and although the gold-bearing gravels were found by the early prospectors to be, in places, only thinly covered by drift, or practically at the surface, yet in most places they are deeply buried beneath glacial deposits. These drift deposits obscure the ancient rock channels, while their thickness and water bearing character have rendered difficult the recovery of the gold. Thus, a considerable part of it still remains. Had the area been un-glaciated, it is probable that practically all the gold would have been recovered within a few years after the discovery of the creeks.

In the early days of mining in the district, the gold was, for the most part, recovered by drifting. Later, this practice gave place to hydraulicing, and some of the hydraulic mines have been in operation nearly every year for the past 40 years. In recent years attempts have been made to interest capital in the possibilities of gold dredging, a phase of mining which as yet has not been attempted in the Barkerville district, although it has been successfully carried on in many other placer regions. The object of this paper is, chiefly, to draw attention to the possibilities of gold dredging on Antler and

(1) Published by permission of the Director of the Canadian Geological Survey.

Williams creeks in the Barkerville area, and to describe the general character of the placers and their associated deposits.

Parts of the Cariboo district were examined geologically by G. M. Dawson in 1876 and 1894. The results of his work, and a summary of the history of placer mining in the region, are given in the Report of Progress of the Geological Survey of Canada, for 1876-77; in the Annual Report for 1887-88, part II; and in the Summary Report for 1894. The Cariboo district was topographically and geologically mapped in 1885 and 1886 by Amos Bowman, assisted by James McEvoy. The first part of Bowman's report, dealing chiefly with the general geology of the district and the possibilities of lode mining, was published in the Annual Report of the Geological Survey of Canada for 1887-88. The second part, in which it was intended that detailed descriptions of the principal auriferous creeks should be given, was not published. A number of detailed maps of the creeks were prepared by Mr. Bowman, but in June, 1894, he died without having written any descriptive matter to accompany these maps. The maps, however, were published in 1895 by the Geological Survey, and were accompanied by an explanatory note by G. M. Dawson.

In 1918 B. R. MacKay, of the Geological Survey, began an investigation of the Cariboo gold region and continued his work during parts of 1918 and 1919. The partial results of his work are given in the Summary Reports of the Geological Survey for those years. In 1920 a topographic map of the Barkerville area, embracing 210 sq. miles, was made by D. Nichols of the Topographical Division of the Geological Survey. The map will be published on the scale of 1 mile to 1 inch, with a contour interval of 100 feet.

Much valuable information regarding mining operations in the district is given in the reports of the Provincial Minister of Mines, which have been issued annually since 1874. The report for the year 1902 is of special interest, as it contains a comprehensive description by Wm. Fleet Robertson, Provincial Mineralogist, of the placer deposits and the mining operations then in progress. In the report for 1914, the mining operations of the district are described by J. D.

Galloway, Assistant Provincial Mineralogist, and a report is now submitted annually by Mr. Galloway as resident engineer of the district. In other years the general progress of, and information about, the district have been given by the gold commissioner.

The history of mining in the district prior to 1878 is given in considerable detail by Bancroft in his "History of British Columbia," published in 1887, and also by F. W. Howay and E. O. S. Scholefield, in "British Columbia from the earliest days to the present". Interesting data on the subject is to be found in numerous other publications and manuscripts, including the early reports of the gold commissioners, which are available for reference in the excellent Parliamentary library at Victoria. A few rare volumes and manuscripts on Cariboo are to be found in the Public Archives at Ottawa, also. These old reports, as well as the later reports and Bowman's maps, are of interest and importance because they furnish the main evidence as to what creeks were thoroughly tested in the early days of mining, and what creeks were not: questions which, because of the peculiar conditions existing in Cariboo — the presence of glacial drift overlying the gold bearing gravels—have arisen many times in the past, and will doubtless arise in the future.

Two papers by J. B. Tyrrell, dealing with the geology and placer mines of Cariboo, have been published recently. One of these, entitled "Was there a 'Cordilleran Glacier' in British Columbia?" appeared in the *Journal of Geology*, vol. 27, 1919, and the other, "Notes on the placer mines of Cariboo, British Columbia", was published in *Economic Geology*, vol. 14, 1919 [1].

The writer spent two months in 1921 in an examination of the placer deposits of the Barkerville area and adjacent parts of Cariboo district, continuing the work of B. R. MacKay. Attention was directed mainly to Williams, Antler, and Grouse creeks, and detailed maps were made of parts of these. No attempt is made here to give a general account of the possibilities of placer mining in the area, for, in the short time devoted to the study of the area, it was only possible to gain a general

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(1) A paper on "Cariboo Placers and Lodes", by J. A. Macpherson, appeared in the Min. and Eng. Record, vol. XXIV, 1919.

knowledge of the history of mining of a few of the numerous creeks, and of the mode of occurrence and origin of the pay gravels.

The writer is indebted to Mr. John Hopp for information regarding the results of Keystone drilling on Williams creek, and to Mr. J. G. McLaren for information regarding the results of drilling on Antler creek. Acknowledgments are due also to Mr. Lawrence Muller, manager of the Hopp mines, to Mr. C. W. Moore, manager of the New Waverly hydraulic mine, to Mr. L.A. Dodd, gold commissioner, to Mr. L.A. Bonner, and to many other mining men of the district.

## ACCESSIBILITY.

The Barkerville area may be easily reached by the Pacific Great Eastern Railway from Vancouver to Quesnel or by steamer from Prince George to Quesnel, and thence 60 miles by stage to Barkerville, or by automobile stage from Ashcroft to Barkerville. The Keithley district is reached by a road passable for automobiles from Williams Lake to Quesnel Forks, and thence 18 miles by a rough wagon road to Keithley. A pack trail connects Keithley and Barkerville, and wagon roads connect most of the important creeks in the Barkerville area.

## PHYSICAL FEATURES.

Because of its considerable relief, the Barkerville area is generally described as mountainous, but, as pointed out by G.M. Dawson[1], it is perhaps more correct to regard the region as a deeply dissected high-level plateau, the plateau level being preserved only on the inter-stream areas. The Cariboo mountain range, characterized by rugged peaks covered with perennial snows, lies about 20 miles east of Barkerville and forms the eastern boundary of the plateau. In the vicinity of Barkerville, the plateau level is 6,000 to 6,200 feet above the sea. Most of the important gold-bearing streams, as the Antler, Grouse, Williams, Jack of Clubs, and Lightning, head in the Bald mountain plateau — so called because it is largely above the timber line—above which rises Mt. Agnes, an isolated peak about 300 ft. above the general plateau level. With

(1) Geol. Surv. Can., Ann. Report, Vol. III, Pt. II, 1887-88, p. 30R.

the exception of some of the tributaries of Swift river, only the creeks flowing north, northeast, and northwest, from Bald mountain are gold-bearing. The creeks descend rapidly from the plateau level so that within a distance of 5 to 10 miles from their sources they flow in valleys, the bottoms of which are in places over 2,000 feet below the plateau level. The valleys are narrow and steep-sided in the upper parts and broaden out in the lower parts, where they are deeply drift filled and have alluvial flats with a general elevation of about 4,000 feet.

The remarkably flat-topped summits on the inter-stream areas, and the fact that the nearly level surface is formed by the truncated edges of a highly folded and steeply dipping series of sedimentary rocks — which were nearly horizontal when deposited — show that the area was once mountainous and was worn down nearly to a plain. The deep valleys have been cut down below the plateau level and are evidence of still further erosion of the bedrock, so that it is evident that several thousand feet of rock have been eroded away in places. Here, then, were almost ideal conditions for the formation of rich placer deposits — very extensive and prolonged erosion of rocks which were in part gold-bearing and the subsequent preservation of the resulting deposits by a thick covering of drift.

The area is well wooded except on the summits, and an abundance of timber, chiefly spruce, is available.

## GLACIATION

A remarkable fact and one difficult of explanation is that, although the region was glaciated, the gold placers were not destroyed or materially eroded. The gold-bearing gravels are in part post-glacial, but by far the most important deposits are pre-glacial or Tertiary in age. This is obvious from the fact that the placer deposits are, largely, buried beneath glacial drift, and because only by the erosion of considerable thicknesses of the bedrock, and concentration of the gold by stream action, could the gold placers have been formed.

Glacial drift occurs abundantly in the area, and in places fills the valley bottoms to a depth of 100 to 250 ft. It frequently mantles the sides of the valleys up to an altitude of over 1,000 feet above the valley bottoms, and in places, as for example on Proserpine mountain, true boulder clay occurs at an altitude of 5,500 feet. At higher altitudes the drift is usually very thin or is absent, but erratics occur, as pointed out by B. R. MacKay[1] on some of the highest summits, at altitudes of nearly 6,500 feet. The glacial drift in the several valleys is nearly all of local origin, but foreign boulders occur. For example,large boulders of conglomerate are fairly abundant in the upper part of Williams creek. Glacial striae are well developed in places. They are well shown on both sides of the valley of Williams creek, a short distance above the canyon between Barkerville and the old town of Richfield. Here the striae trend downstream and show that the bedrock of the sides of the valley was slightly eroded by the glacier; but the valley bottom was not eroded by the ice, for very rich placer ground occurred along this part of the valley bottom. A considerable area of bedrock, glacially striated and grooved, is exposed by mining on Cunningham creek. It is stated that large amounts of gold were obtained from the creeks and from fissures in the bedrock at this locality, showing that erosion by the glacier was not sufficiently great to entirely remove the pre-glacial deposits. Striae also occur at other places in the area. They all, with possibly one or two exceptions, trend downstream in the several valleys and were therefore probably formed by valley glaciers. Moraines formed by the valley glaciers occur, as has been pointed out by J. B. Tyrrell [2], in Lightning creek valley near Stanley, and in Slough creek and Grouse creek valleys. These moraines were formed during the closing stage of glaciation, for they are well preserved and occur in the valley bottoms. There is, therefore, as was recognized by Tyrrell [2], comparatively little evidence of the presence or effects of an extensive ice-sheet in this region during Pleistocene time. Nevertheless, the presence of erratics at high altitudes, the foreign derivation of some of the drift in the several valleys, and

(1) Geol. Surv. Can., Summary Report 1918, p. 42B.
(2) "Was there a 'Cordilleran Glacier' in British Columbia?". Jour. of Geol. Vol. 27, 1919, p. 57.

the occurrence of boulder clay at altitudes too great to be referable to valley glaciers, all go to show that an ice-sheet of considerable thickness did exist.

The glacial drift consists in part of boulder clay and moraine material, in part of stratified sands and silts — the 'slum' of the placer miner — and in part of stratified gravels and bouldery deposits. The boulder clay is in most places divisible into two parts, an upper yellowish unstratified clay containing comparatively few boulders, and a lower bouldery part, occurring chiefly in the valley bottoms. It is probable that the upper part was formed from materials included in the ice and deposited after its melting, and that the lower part was formed beneath the ice. The moraines are composed of materials deposited at the ends of the glaciers. In many places the stratified sands, silts and gravels occur chiefly beneath the boulder clay and rest directly on the pre-glacial gravels and 'slide-rock' of the valley floors. The stratified silts and sands above the general drainage level in some of the valleys and beneath the boulder clay, show that at an early time during the period of glaciation the northward drainage was blocked, probably by an advance of glaciers from the Cariboo mountains, and that glacial lakes were formed in the area. The deposition of the sands and gravels protected the pre-glacial gravels and the bedrock from erosion by the glaciers that subsequently covered the area.

The main reasons, therefore, why the placers were not eroded by the glaciers appear to be that they were partly protected by glacial outwash deposits, and that the ice-sheet which covered the whole region was nearly stagnant. If valley glaciers alone had persisted throughout the ice-age, it is probable that the pre-glacial gravels would have been entirely removed, for it is well recognized that valley glaciers are much more effective agencies of erosion than is an ice-sheet of wide extent that has a low surface gradient. Valley glaciers did exist at the close of the ice-age and they accomplished some erosion, especially in the upper part of the creek valleys. This probably accounts, in part, for the absence in places of gold values in the gravels.

An interesting feature in connection with the glaciation of the area is that, so far as known, only the one boulder clay or till sheet occurs and, as pre-glacial gravels are known to occur, it is probable that, so far at least as the higher parts of the region are concerned, the glacial period was a unit.

## THE PLACER DEPOSITS.

The placer deposits may be broadly divided into post-glacial and pre-glacial gravels, the latter being by far the more important. The post-glacial gravels form the surface gravels (except where buried by tailings) in the valley bottoms and in a few places on benches. Their gold content was derived by stream erosion of the glacial drift, which contains small amounts of gold included in it, and from erosion of the pre-glacial gravels and bedrock into which the present streams have cut in a few places. The area is largely drift covered, especially the valley bottoms, and as no large amount of erosion has taken place since the disappearance of the glaciers, it is evident that the post-glacial gravels could not be of much importance, except locally.

The pre-glacial gravels consist of bench and creek gravels largely buried beneath the drift deposits, but occurring, in places, nearly at the surface. They are the source from which the greater part of the output from the Barkerville area has been derived. It is by no means easy to distinguish the pre-glacial from the post-glacial gravels, even where good sections are exposed down to the bedrock, for both consist mainly of gravels derived locally, and both may be well oxidized or not. The pre-glacial gravels are for the most part only a few feet in thickness, and in places they are distinguished by the presence in them of the heavy iron minerals, and of pebbles or boulders of galena and barite, and occasionally scheelite. A characteristic of many of the deposits is the presence in them of large angular slabs of rock known as 'slide rock'. It is, apparently, ancient talus; and blocks of it are partly embedded in disintegrated bedrock. In places the gold extends down for several feet along the bedding planes and fissures in the bedrock. There are no large quantities of residual gravels,

that is, gravels consisting principally of a resistant mineral such as quartz. Usually, they are made up of fragments of various kinds of schist, limestone quartz, and igneous rock, material that has been derived mainly from the Cariboo schist series of rocks as named and described by Bowman [1].

Bench deposits occur on many of the creeks at various levels above the creek bottoms. They are nearly all confined to the present valleys and rarely trend across the present drainage; for, as pointed out by G. M. Dawson [2], the ancient stream courses pursued very much the same general directions that their present representatives follow. There are, however, many places where diversions and even reversal of stream drainage took place as the result of stream capture in pre-glacial time. In places, also, the irregular deposition of glacial drift has diverted the streams into new channels, a striking example of which is Black Jack canyon on Williams creek. A characteristic feature of several of the creeks is that the new channels, indicated by their narrow steep-sided character, rarely contain much gold. Some of these are mainly post-glacial in age, others are pre-glacial and are partly due to stream capture and possibly also to the rejuvenation of the streams by uplift of the region. They contain little gold unless they happen to have been cut down directly beneath the older higher channels, for the concentration of gold in them is the result of the erosion of a small thickness of rock, whereas the concentrations in the old channels are the result of the wearing down of great thicknesses of rock. The gold in the area is mostly coarse, so that little is actually transported by the streams. Part of it has probably moved downwards, vertically, nearly as far as it has been transported horizontally. Where streams flow through canyons, their current is rapid, and any gold fine enough to be transported by the stream would probably have been carried through the canyon.

The gold in the placers in the Barkerville area is mostly coarse. Very little flour gold or even fine scale gold is present. Apparently the streams in pre-glacial time had sufficient gradient to transport the fine gold out of the region to the lower Fraser, where it has gone to enrich the placers of that region,

(1) Geol. Surv. Can., Ann. Report, Part C, 1887.

(2) Geol. Surv. Can., Ann. Report Vol. III, Part II, p. 31R.

There is little difficulty, therefore, in saving the gold in the Barkerville district.

In places, masses of the pre-glacial gold-bearing gravels are included in the boulder clay, especially in its lower, bouldery part. The upper, clayey part is almost free from gold, as are also the stratified sands and silts and fine gravel known to the miners as "chicken feed"; but occasional nuggets are scattered through the glacial drift. There have been some notable instances in the district of the finding of masses of gold-bearing gravels included in the boulder clay, but, because of their mode of occurrence, such masses are usually found only accidentally in hydraulic mining. In places where the boulder clay is tight on the bedrock, and contains little gravel, there is little gold. This point was well recognized by the early miners, and isolated masses of boulder clay in the bed of the upper part of Williams creek remain untouched to this day, although the ground all around them was worked. In the upper parts of several of the creeks considerable quantities of coarse bouldery deposits, in places rudely stratified, occur. They are apparently glacial outwash deposits and are mostly nearly barren of gold.

An instance of the sporadic occurrence of gold in the drift deposits of the area was noted during the past summer. Two miners, George Mcore and Joseph Spratt, working on the upper part of Walkers gulch, which empties into Williams creek at the old town of Richfield, recovered 7 ounces of gold in 3 days' hydraulicing. The material washed was largely boulder clay, and it averaged over 3 dollars a cubic yard, the gold being derived apparently from gravels included in the boulder clay. There was very little gold on bedrock. It is probable that there are many such occurrences in the district.

## ANTLER CREEK

Antler creek was discovered in 1860 and was the first creek of the Barkerville area found to be auriferous. The rich placers of the Barkerville area were not found by following the trail of fine gold up the Fraser and its tributaries directly to this area. However, the theory of the miners that this fine gold had its source in a region where coarse gold would be found was

undoubtedly the impelling force, as has been frequently pointed out, that led to the rapid discovery of the main placer fields. The theory was more or less sound in respect to the discovery of Keithley, Harvey, and Cunningham creeks, southeast of the Barkerville area. In 1859, while placer mining was being actively carried on at various places along Quesnel river as far as the Forks and Quesnel lake, a number of prospectors made their way up the north fork of the Quesnel, but lost the trail of fine gold at the falls, 5 miles below Cariboo lake; and it was not until the following year that Keithley creek, flowing southeast into Cariboo lake, was discovered. Late in the autumn of 1860 a party of prospectors, among whom were Rose, an American, and McDonald, a Canadian, crossed over the divide at the headwaters of Keithley creek and discovered gold on Antler creek. Little work was, apparently, done on the creek in 1860. Rumors of the success of the miners becoming known at Keithley, a mid-winter rush to the creek ensued and when gold commissioner Nind, who was called to settle mining disputes there, arrived at Antler creek early in March, 1861, he found the snow 6 or 7 feet deep and the miners living in holes which they had dug in the snow and subsisting on the scanty supplies brought in from Keithley [1].

Antler creek rises in Bald mountain plateau, flows southeast for several miles to Sawmill flat where it is joined by a tributary from the south, and there turns and flows nearly north to form the headwaters of Bear river. Four miles north of Sawmill flat the creek turns to the left around a hill, to the right of which an old channel, partly drift filled, extends to the headwaters of a branch of Cunningham creek in Cunningham pass. This old channel is about 80 feet above the bedrock in the present stream channel of Antler and is a continuation of a bench which extends up-stream along the left limit of Antler creek for about 3,500 feet and then crosses to the right bank. Above this point a small canyon extends for nearly a mile to the wide shallow part, where most of the mining in 1861 was done. The remarkably rich and easily mined part of Antler creek was shallow ground extending for about 1½ miles

(1) Bancroft's Works, Vol. XXXII, History of British Columbia, pp. 472-494.

along the stream above the canyon and below Sawmill flats. It is stated that the daily aggregate gold output of the creek during the summer of 1861 was valued at over $10,000 and that much of the ground yielded $1,000 to the square foot [1].

The benches along the creek below the canyon and down as far as Cunningham pass were mined at various places and at various times down to the present, partly by drifting and partly by hydraulicing. It was early recognized by the miners that the present stream gravels would be enriched at those places where the present stream valley has been lowered by erosion and has cut across the more ancient stream valley, represented in part by the benches. One of these places, as already pointed out, was at the mouth of the valley known as Whiskey flat, leading to Cunningham pass. Here the Nason Company sank two shafts and attempted to drift out into the channel. It is stated that prior to 1875 the Nason Company expended $30,000 without succeeding in testing their ground. Work was again started by the Company in 1884 and was carried on nearly every summer until 1893, but with little success because of the porous, water-bearing character of the river gravels and the absence of clay which would serve to keep out the surface water. In 1888 they succeeded in driving part way across the creek bottom, and obtained round, heavy gold at the rate of $1.50 to the bucket; but they were of opinion that they had not struck the main lead [2].

The dredging ground of Antler creek consists of the Nason (real estate) claim; the Lothair (real estate) claim, extending up-stream above the Nason to the narrow canyon-like part; and the Macdonald lease, below the Nason. It embraces the comparatively shallow part of the present stream bed, which begins to widen out in the lower part of the Nason claim. Below the Macdonald lease, the stream bed is wide and deep, but possibly contains dredging ground. Keystone drilling, to determine the depths and gold values of the ground, was done by the Yukon Gold Company in 1915. The results of the drilling showed that the known dredging ground has a length of 4,000 feet, an average width of 140 feet, and an average depth

(1) Bancroft's History of British Columbia, p. 492.
(2) Ann. Reports, Minister of Mines, B. C., 1875-1893.

of 48 feet, giving a total of 1,000,000 cubic yards. The average value per cubic yard is 66.3 cents and the total recoverable gold content is estimated at $540,000. The upper 1,250 feet of the property has not been drilled. In this part the values should approximate the general average of the drill results, and should bring the total gross recoverable gold up to $700,000. From the fact that the present channel cuts across the old gold-bearing channel in the Nason claim, and also in the upper part of the Lothair claim, and is thereby enriched, it is probably safe to infer that the ground will dredge up to the drillings. The gravel is mostly fine, with few boulders over 6 inches in diameter, and clay is absent. The absence of clay, in fact, has been the chief reason why it has not been found possible to mine the bedrock gravels by drifting.

### WILLIAMS CREEK

Williams creek, on which the town of Barkerville and the old town of Richfield (a mile up-stream from Barkerville), are situated, heads in Bald mountain and flows north for 5 miles to "The Meadows", where it turns west and forms the headwaters of Willow river. The gold-bearing part of the creek, extending for 1¾ miles above Barkerville and 1 mile below, is naturally divided into two parts, of different characteristics, by the Black Jack canyon, about half-way between Barkerville and Richfield. The canyon is narrow and rock walled, about 50 feet deep and 1,000 feet long. It forms a bend in the stream around a rock island, on the east side of which the pre-glacial stream flowed in a narrow valley. The filling of the old valley by glacial drift caused the present stream to be diverted into the new channel. This fact was recognized long ago by the miners and as early as 1862, one year after the discovery of the creek, the Black Jack tunnel was begun for the purpose of locating the old channel. In 1867 the property was worked as a hydraulic claim, the first to be started in the area, and it was still being worked in 1890 [1]. The drift deposits were eventually largely washed out from the old channel and at the present time the stream flows partly in one and partly in the other channel.

(1) Ann. Report, Minister of Mines, B.C., 1890, p. 360.

Discovery claim on the creek is at the lower end of the canyon, and was located in 1861 by William Dietz after whom the creek was named. It proved to be one of the poorest claims on the creek [1].

The part of Williams creek valley above the canyon is comparatively narrow and steep sided. The first important discoveries of gold were made in this part of the creek, between the canyon and Richfield. The post-glacial gravels were first worked, the underlying boulder clay being regarded as the bedrock; but the main 'pay' was soon found on the Abbott claim by sinking through the boulder clay to the bedrock. It was probably the comparatively lean character of the upper gravels which led to the first naming of the creek as Humbug creek. The main 'pay' on the Steele claim consisted of a blue clay layer 6 feet thick containing decomposed slate and gravel, the overburden being 8 to 18 feet thick [2]. This part of the creek was eventually worked by open-cuts from the surface to the bedrock. A poorly defined bench 50 to 100 feet above the creek bottom occurs along the east side from the canyon to a short distance above Richfield, and was also worked. The upper part of the creek was found to be barren; the pay stopped abruptly about three-quarters of a mile above Richfield, but was again met with considerably higher up, in the tributary valley, McCallum's gulch [3].

The lower part of the creek below the canyon is comparatively wide and flat-bottomed. Here the ground is deep and was mined by sinking shafts to depths of 40 to 80 feet and removing, by means of drifts, the pay gravels on bedrock and washing them at the surface. The first shaft sunk in this part of the creek is still to be seen in the town of Barkerville. It was put down late in the season of 1861, or early in 1862, by Billie Barker, after whom the town was named. As in the upper part of the creek, here also there were two pay streaks, one in the post-glacial gravels and the other in pre-glacial gravels on bedrock. The latter are separated from the upper

(1) Bancroft's History of British Columbia, p. 497.

(2) Brancroft's History of British Columbia, p. 496.

(3) Bowman's map of Williams Creek, Geol. Surv. Can., Annual Report, 1895.

gravels by glacial deposits consisting in part of boulder clay. The post-glacial gravels are largely concealed by tailings, which average possibly 20 ft. in thickness, and were chiefly derived by hydraulicing in Stout's gulch where it enters Williams creek from the west at the lower end of the canyon. The pre-glacial gravels were mined chiefly in the '60's and were largely worked out in that part of the creek extending from the canyon down nearly to the meadows, where the excessive pressure of the ground water prevented mining. These gravels were remarkably rich. According to the estimate given in the Report of the Provincial Minister of Mines for 1896, this part of the creek bottom produced between $8,000,000 and $9,000,000 in gold. The post-glacial gravels were also found in places to be rich. At the famous Cameron claim, about half a mile below Barkerville, the ground was sufficiently rich to be worked from the surface down to bedrock [1].

The most important tributary of Williams creek below the canyon is Conklin gulch, which enters from the east opposite the town of Barkerville. It was mined throughout a distance of nearly 2 miles, partly by drifting and partly by hydraulicing, the ground varying in depth from 20 feet in the upper part to 90 feet in the lower [2]. The famous Ericcson and Sawmill claims were located in the lower part, near Williams creek, and the equally famous Aurora was on a bench on the right side, near the mouth. On the right limit of the gulch, near its mouth, are two benches, above the level of Williams creek flat, and the present stream flows between rock rims nearly at the level of the flat. The deep channel, graded to the bottom of Williams creek valley, is buried beneath drift. The three claims mentioned above are credited with having produced $1,500,000 in gold [3].

The Forest Rose property, which was originally mined by a deep shaft and by drifting, and was later worked as a hydraulic mine for nearly 30 years, occupies a buried channel running nearly parallel to Williams creek in its lower part and lies a short distance to the east of this stream. The first hydraulic work

(1) Milton and Cheadle. The Northwest Passage by Land, p. 362.

(2) Dawson, G.M., Geol. Surv. Can., Report of Progress. 1876-77, p. 135,

(3) Ann. Report, Minister of Mines, B.C., 1896, map of Williams Creek.

on the property began at the St. George's (originally the Prairie Flower) cut at the meadows. The bedrock was, however, not reached until after several years' work and not until the cut had been carried for some distance up the old channel [1]. The old rock channel is wide, has a low gradient, and is about 50 feet above the bedrock in the adjacent part of Williams creek. It is evidently an old channel of the latter, as is generally recognized by the miners, and it is not, as was formerly held, a high-level channel extending along the east side of Williams creek from Conklin gulch [2]. The channel apparently leaves Williams creek at about the location of the old Cameron claim.

Attempts have been made, notably by the Cariboo Gold Fields Company, using a hydraulic lift and bucket elevator, to mine the whole of the gravels of the part of Williams creek below the canyon; for it was generally held that, because of the crude methods of mining and of saving the gold at the time when nearly all the mining was done on this part of the creek, and because of the large amounts of gold recovered, there must be a considerable amount remaining. These attempts, however, failed owing to mechanical difficulties in the operation of the lift and elevator [3]. In this connection, the following observation of G. M. Dawson [4], is of particular interest: "I do not think it would be an extravagant statement to say that the quantity of gold still remaining in the part of Williams creek which has been worked over, is about as great as that which has already been obtained."

The dredging ground on Williams creek extends from the lower end of the canyon or mouth of Stout's gulch to the meadows, a distance of 8,000 feet. The ground was tested by Keystone drilling in 1914 and 1915, eight lines of bore holes being put down at fairly regular intervals across the valley.

The results of the borings showed that the 'pay' averages nearly 240 feet in width and has a maximum depth of 76 feet with an average of 59 feet. The average value is 31 cents per cubic yard, which would probably have to be cut down by 2

(1) Ann. Report, Minister of Mines, B.C. 1886, p. 196.

(2) Bowman's Map of Williams creek, Geol. Surv. Can., 1895.

(3) Ann. Repts., Minister of Mines, B.C., 1899, p. 621 and 1903, p. 60.

(4) Report of Progress, Geol. Surv. Can. 1876-77, p. 114.

or 3 cents to allow for gravels sliding into the pond and other factors. The total yardage of the pay gravels is approximately 4,000,000 cu. yd. Owing to lack of borings, there is some doubt as to the yardage and values in the part of the creek above Barkerville.

For the reasons stated above it is surprising that the results of the drilling did not show higher values, and it seems possible, when the ground is actually dredged, that in places it will produce higher values. It is possible, also, that dredging ground may be found at the lower end of the Forest Rose and in places in the Williams creek flats. The old channel of the Forest Rose is known to have been rich ground, for it is credited with having produced, prior to 1896, $480,000, in addition to which there was an output of $100,000 from the Prairie Flower [1]. Possible objectional features of the Williams creek dredging ground are the depth of the deposits; the presence of timbers in the old underground workings and in the bedrock drain, which was built in 1866 and extends throughout the area; and the presence, in places, of boulder clay. It is possible that much of the underground workings has caved and that the timbers, like those excavated in the Gold Fields workings in the lower part of the creek, are somewhat shattered. The gravel deposits contain few large boulders and the boulder clay occurs only in places. On both Antler and Williams creek the bedrock consists of steeply-dipping schists that are easily excavated. It is possible that a considerable part of the tailings, which form a nearly barren overburden, might be removed by "booming". An excellent site for a dam is at the lower end of the Black Jack canyon, in which large volumes of water could be impounded and the water released automatically to flush out the tailings. The creek, from the lower end of the canyon to the Meadows, has a surface gradient of 2.6 per cent. During the past few years the creek itself has cut the tailings in places as much as 4 feet. The lowering of the creek bed at Barkerville, which is protected from creek floods by a bulkhead, would also make it possible to work Conklin gulch to the best advantage, either by hydraulicing or by dredging; for, as already pointed out, the ground is deep in the lower part near

(1) Ann. Rept., Minister of Mines, B.C., 1896, Map of Williams creek.

Williams creek and this was the richest part of the gulch.

There is, possibly, dredging ground on Lightning creek, and at other places in the district also. The writer's investigation, however, was confined chiefly to the areas which have been described above.

## SUMMARY.

Gold dredging has not as yet been attempted in the Barkerville district, although it has been well-known for several years that ground suitable for dredging occurs in the area. One of the reasons for this failure to dredge is the present high cost of machinery, of supplies in general, and of transportation. There is, however, a probability of dredging being started in the near future on Antler creek and possibly, also, at other places in the region. The results of Keystone drilling on Antler and Williams creeks have shown that, in parts of the creeks, gold to the value of at least $1,800,000 exists, of which the great part is recoverable. In this paper, an attempt has been made, by reciting the history of mining on the creeks and describing their physical characteristics, to show that the estimates of gold values, as evidenced by the borings, are moderate, and that it is possible the known dredging ground may be extended.

# SOME ORE DEPOSITS OF THE COAST RANGE.

By W. M. Brewer.

Annual Western Meeting, Vancouver, November, 1922.

In the notes that follow, I have confined myself to that portion of the Coast range extending from the Fraser River valley to Seymour inlet at the south end of Queen Charlotte sound. In this territory are included New Westminster and Vancouver Mining Divisions as well as the Mainland portion of the Nanaimo Mining Division, the whole forming the eastern, northern and north-eastern parts of the Western Mineral Survey District, or No. 6 District. The northerly extension of the Coast range from Seymour inlet is in the North-western Mineral Survey District, under the administration of Mr. George Clothier as Resident Engineer, and naturally my personal observations during recent years have not been extended beyond the boundaries of my own district.

Because of the great extent of the Coast Range granodiorite batholith in my district, I have been particularly interested in the examination of this great complex, with the idea of encouraging prospecting therein; and I have prepared the following paper with that end in view as my chief motive.

The early history of mining in British Columbia was a repetition of the early history of metalliferous mining generally throughout the world. The earliest discoveries were of placer gold and were made by the old-time prospectors of the middle of the last century; but although placer gold discoveries were made on the Fraser river near the coast as early as about 1858, and a little later in Cariboo, it was not until about 1870 that these discoveries were followed by mineralized bodies, lodes, ledges, or other solid deposits, carrying values in either precious or base metals, being found in that range of mountains designated by the late Dr. G. M. Dawson as the "Coast range" to distinguish it from the lava-built Cascade range as originally named at the Columbia river. Dawson's name for the range

has survived and is, in fact, the only name officially approved by the Geographical Board of Canada for any principal division of the Pacific cordillera.

## GEOLOGICAL REPORTS.

Aside from occasional references to be found in Dawson's earliest reports, written about the year 1875, practically the only Geological Survey reports on the Coast range are as follows:

Geology of the Coast and Islands between the Strait of Georgia and Queen Charlotte Sound, B.C., by J. Austen Bancroft; Canadian Department of Mines, Geological Survey, Memoir 23, 1913.

Report by Charles Camsell on his reconnaissance along the line of the P. G. E. Ry.; published in the Geological Survey Summary Report for 1917.

Report by J. S. Schofield on the Britannia Map Area; published in the Geological Survey Summary Report for 1918.

In describing the extent of the Coast Range, Bancroft says:

"West of the Interior Plateau, the rugged Coast range borders the Pacific ocean for a distance of 900 miles, extending in a N.N.W. direction from the Fraser River valley on the south to the head of Lynn canal. It continues northward, passing behind the St. Elias range of Alaska, where it gradually blends with the Interior Plateau at lake Kluane in longitude 138° 30′ near the boundary line between the Yukon Territory and Alaska. In southern British Columbia these mountains maintain a width of about 100 miles, which diminishes to about 50 miles at the Lynn canal. It is not, however, easy to assign definite boundary lines to them, for they shade off into Interior Plateau conditions on the east, and are closely related in their geological and physiographic developments to many of the islands on the Pacific coast. The length and breadth of this range is coincident with the main portion of the great series of batholiths, composed of granites, diorites, gabbros, etc., of remarkably uniform texture, which were intruded probably during late Jurassic time."

## GEOGRAPHY.

The geographic boundaries of that portion of the Coast range to be described are roughly as follows:

The southerly side of the entrance to Seymour inlet forms the northwestern limit. From here the boundary follows a northeasterly direction along an irregular line representing the summit dividing the waters which flow unto Queen Charlotte sound, north of Seymour inlet, from those which flow to the coast south of that inlet. The northeasterly corner is approximately the source of the Klina Klina river, near the northwesterly boundary of that part of the Interior Plateau known as the Chilcoten country. The easterly boundary follows the irregular summit dividing the waters which flow easterly into the upper Fraser river from those flowing westerly and southerly to the coast or into the lower Fraser river west of the old town of Hope. The southeasterly corner is at the International boundary near the east end of Chilliwack lake. On the south and west, the area in bounded by the International boundary and the coast line, respectively, the numerous islands lying between the coast and Vancouver Island not being included.

Reference to a map of the area here outlined will show how numerous are the inlets or fiords which penetrate the mainland for several miles, cross-cutting the grano-diorite batholith, and thereby affording prospectors most excellent opportunities for examining the mountains at many different locations, without the necessity of having to convey supplies overland from the railways near the southerly boundary through a heavily timbered mountainous country, in which many of the peaks reach an elevation of from 6,000 to 8,000 feet above sea-level. The inlets are all navigable for both large and small boats. Vessels drawing 30 feet or more can be navigated with ease and safety, while the waters are so well sheltered that prospectors can with perfect safety use gasolene launches, row-boats, or canoes to convey supplies to central points from which to start exploration.

From south to north, the most important of these inlets are: Burrard, Howe Sound, Seechelt, Salmon Arm, Narrows Arm, Jervis, Hotham Sound, Malaspina, Theodosia Arm,

Toba, Bute, Frederick Arm, Phillips Arm, Loughborough, Knight and Seymour.

These various inlets are all fed by rivers of considerable length, most of them torrential streams which, through much of their course, flow, with many rapids, through canyons, but which are generally navigable for canoes for varying, though comparatively short, distances above their mouths. The longest of the rivers are the Klina Klina and the Homathka; the former, about 70 miles long, empties into Knight inlet, while the latter is about 65 miles long and empties into Bute inlet.

In addition to these natural cross-sections, the Coast range, in its southern part, is also cross-cut by the grades of the Canadian National, Canadian Pacific, and Pacific Great Eastern railways.

Despite the several arteries made by nature as well as by man, and the excellent opportunities thus afforded for exploring the mountains, it has only been within the past twenty-five years or so that the Coast range has been even occasionally visited by prospectors; and even up till quite recently the section has not appeared to prove the attractive field for prospecting which its early history, to be narrated later, and the geological conditions, as described by Charles Camsell, would indicate it should be.

It is easy to understand why the discovery of placer gold on the Fraser river about the year 1858 — when the late Jim Moore and partners discovered "pay" on Hill's Bar near Yale — was not followed by a general prospecting for quartz ledges through the mountainous Coast range. The prospectors of those early days were only looking for placer gold. They neither cared about lode mining nor understood how any other minerals than gold dust or nuggets could be made available. In other words, ores, to them, were useless, since they were not able to distinguish the several varieties, or, even if they could recognize them, they could not determine their contents. Besides, after outfitting at Victoria or New Westminister, they travelled up the Fraser river, and would only be too pleased to reach such camps as Yale, or at least Hope, *en route* to the upper river where bars carrying good "pay" had already been located.

Consequently, there is no record of any locations of lode minerals being made until early in the seventies.

## EARLY HISTORY.

The earliest recorded prospecting in the Coast range was near the head of Salmon arm of Seechelt inlet, where bornite was discovered in the early seventies. The prospects, known as the "Howe" mine, were worked for a time, the ore being brought down the mountains by Indians and some of it shipped to Swansea. The next prospect recorded was on the easterly boundary of the grano-diorite batholith, near the town of Hope, at an elevation of about forty-five hundred feet. The ore, carrying silver values principally, occurs very near the contact between the granite rock of the Coast range and the older sedimentaries of the Interior Plateau. Some ore from this prospect, known as the "Eureka" mine, was also shipped to Swansea. Both these groups of mineral claims were owned by Victoria syndicates.

Thereafter, there is no record of prospecting in the Coast range until 1897, when a party of trappers discovered copper ore on Britannia mountain, about three miles from Howe sound. Another party discovered copper ore on Hidden creek, which empties into Observatory inlet, an arm of Portland canal, about six hundred miles in a northwesterly direction from Howe sound.

Following this, prospecting operations were carried on in the vicinity of Shoal bay, Phillips arm and Frederick arm, distant about 130 miles north-westerly from Vancouver. In this section gold-bearing quartz veins were discovered, occurring in bands of highly altered stratified rocks. The "Dorothy Morton", "Douglas Pine," "Alexandra," and "Blue Bell" groups represent some of the prospects on which a considerable amount of development work was done, and from the "Dorothy Morton" about one hundred thousand dollars in gold and silver bullion was extracted in eleven months, between December, 1898, and November, 1899, when the property was closed down. Work has never been resumed

From the foregoing list of prospects, it will be seen that the earliest discoveries of lode minerals were made near the

easterly and westerly borders of the grano-diorite batholith, but it does not appear as though very much systematic prospecting followed the results achieved at the "Eureka" and "Howe" locations. Indeed, it would seem that, even after the discovery of the Britannia and Hidden Creek mineralized belts, the true conditions with regard to the geology of the Coast range were not recognized, even by geologists, until Camsell made his reconnaissance along the line of the Pacific Great Eastern railway in 1917.

## GENERAL GEOLOGY.

In his report, Camsell describes the conditions as follows:

"Geological maps of the western part of British Columbia show the Coast mountains to be made up of a great mass of eruptive rock, mainly grano-diorite, forming a belt from 60 to 100 miles wide and 1,000 miles or more in length. Investigations of the past season have shown that in the section across the Coast mountains along the Pacific Great Eastern railway such a simple condition does not exist, but instead there are several broad bands of stratified rocks — schists, quartzites, argillites, and limestones — which lie in the grano-diorite and have been intruded by it. These bands are remnants of the old cover of the grano-diorite batholith, which were not destroyed at the time of intrusion nor removed by later erosion. They all strike parallel with the trend of the range, and, consequently, their greatest dimension is in a northwesterly direction and their shortest along the line of railway. The contacts of these bands of stratified rocks with the grano-diorite are all more or less mineralized. The lateral contacts are mineralized mainly by copper ores, and on the roof contacts, that is, where the batholith plunges northwesterly or southeasterly under a roof of stratified rocks, the mineralization is often by the more precious metals.

"The significance of the discovery that large bands of sedimentary and other stratified rocks lie in what was formerly supposed to be nothing but a mass of grano-diorite in the Coast mountains is far-reaching, and it is only fair to assume that the other sections through these mountains northward to the Yukon boundary will reveal the same conditions to a

greater or less extent. The conditions are most favourable for the occurrence of metallic deposits of copper, lead, zinc, gold, and silver, and the whole interior of the Coast mountains of British Columbia, therefore, becomes an excellent field for prospecting, and not only the eastern and western borders, as was formerly believed to be the case."

Accompanying Camsell's report is a diagram showing the geology along the route traversed by the Pacific Great Eastern railway between Squamish at the head of Howe sound and Lillooet on the Fraser river. It forms a most useful guide for the prospector who proposes to explore the Coast range.

S. J. Schofield's report on the Britannia area (Summary Report of the Geological Survey for 1918), describes the geology of a section of the Coast range starting from the east side of Howe sound, at Britannia Beach (about 6 miles south from Squamish), and extending easterly towards Indian river, which has its source in the Coast range, about 12 miles south-easterly from Squamish, and empties into the head of the North arm of Burrard inlet.

The general geology of the Indian River section is also described by Camsell in the Summary Report for 1918, and quoted later in this paper.

The section between the Britannia area and Squamish is underlain along the coast line by massive granite, and the same rock extends for about 5 miles easterly from Squamish, or to Ray creek, as observed by myself during several trips along the trail which parallels the Stawamus river that empties into the head of Howe sound, and of which Ray creek is a tributary flowing into the Stawamus river from the east, where the creek has its source amongst high rugged mountain peaks. Ray creek practically cross-cuts a wide belt of schists and argillites which are heavily mineralized with pyrite and some chalcopyrite. This belt of schists is apparently an extension of a similar belt on Indian river, which is described in Camsell's report and referred to later in this paper.

The general geology of the Britannia area is described as follows by Schofield:

"The mineralized zone of the Britannia area is entirely located in a belt of metamorphosed sedimentary and igneous

rocks which forms an inclusion in the grano-diorite batholith of the Coast range. The trend of the rocks in this large inclusion, which has an approximate width of 2 miles and a length of 7 miles, is almost east and west with an average dip of 70 degrees to the south. Hence this mass has a trend which varies about 60 degrees from the general trend of the Coast range. Mr. Camsell also has noted the same phenomenon along the section of the Pacific Great Eastern railway from Squamish to Lillooet.

"The geological horizons represented are as follows:

| Era | Period | Lithological characters |
|---|---|---|
| Quaternary | Recent and Pleistocene. | Gravels and sands. |
| Tertiary | .... | Lamprophyre dykes. |
| Early Mesozoic. | Jurassic? | Aplite dykes.<br>Coast Range batholith.<br>Grano-diorite. |
| | Triassic? | Quartz porphyry and diorite porphyrite.<br>Britannia sills. |
| Palaeozoic? | Devono-Carboniferous? | Britannia formation, argillites (slates), argillaceous quartzites, quartzites. |

"*Britannia Formation.*— This formation consists essentially of dark brown to black argillites (slates) which weather rusty brown. Inter-bedded with these argillites are bands of argillaceous quartzite and true quartzite. The age of these rocks is not definitely known, but LeRoy considers them to be Palaeozoic.

"*Britannia Sills.*— The Britannia sills, which are about 1,800 feet thick, are tabular masses of igneous rock which con-

form in dip and strike with the enclosing slates. The composition of these masses ranges from a quartz porphyry to a diorite porphyrite. The quartz porphyry is a brownish grey rock in which crystals of quartz and orthoclase are embedded in the fine-grained groundmass. The diorite porphyrite is a dark green rock showing crystals of hornblende. The two varieties occur in the same sill. It has not been observed whether there is any definite arrangement of these varieties, but this problem is being studied. These sills are economically the most important members of the rocks exposed at Britannia, since the ore-bodies occur as impregnations in the quartz sericite schist which is a sheared porphyry.

"*Coast Range Batholith.*— The Britannia sills and the Britannia formation, which may be considered as a unit since they were affected by the same geological processes, are surrounded, and underlain at some depth, by grano-diorite of the Coast Range batholith. This great batholith extends from the International Boundary line on the south in a north-northwesterly direction, to Alaska. The main mass holds many inclusions of all sizes which are termed roof-pendants or roof-remnants and are more or less mineralized. Such an inclusion is the area of sedimentary and igneous rocks at Britannia. The rocks of this batholith vary in composition from a gabbro to a granite. In the neighbourhood of Britannia, the batholithic rock is a biotite grano-diorite rather coarse in texture and of a greyish white colour. The contacts of the grano-diorite with the slates and the porphyries are quite sharp and many blocks of these latter rocks are embedded in the grano-diorite.

"*Aplite Dykes.*— Aplite dykes which are associated with the grano-diorite are not numerous and were seen only on the eastern ridge of Sky Pilot mountain. They are white in colour and are made up essentially of quartz and feldspar.

"*Diabase Dykes.*— These dykes are narrow, having a width of from 2 to 8 feet. They are fine grained and dark brown in colour. They consist of glassy feldspar in lath-like individuals with the interstices filled with pyroxene, the latter mineral being in small amount. The small, vug-like cavities which occur in this rock are filled or lined with bituminous material with a spherulitic structure. These dykes are very fresh and cut across the shear planes of the mineral zone.

They may be associated with either the Tertiary or the Pleistocene volcanic field which occurs just to the north of Britannia.

"*Quaternary Gravels and Sands.*— The lower slopes of the mountains in the Britannia area are covered with a variable thickness of glacial drift. Numerous erratics were observed on the summits at an elevation of 5,000 feet. It might be mentioned here that the elevation of the glacial amphitheatre is as low as 4,000 feet. Recent gravels and sands are being deposited at the mouth of the streams.

"*Britannia Shear Zone.*— Mineralization is confined to the great shear zone, sometimes called the Britannia mineral zone, which stretches at least from the Daisy mineral claim on Britannia creek to the Reggie claim on the west fork of Seymour creek, a distance of at least 5 miles. It is certain that the extent is greater than that given above, but the Reggie claim is as far east as the explorations were carried. The width of the shear zone varies greatly, but the maximum width would be about 500 feet. The shear zone strikes 140 degrees (astronomic) and dips from 40 to 70 degrees to the southwest. In the vicinity of the Empress claim the strike changes to 170 degrees (astronomic), but resumes the old strike on the Victoria and Queen claims. An examination of the bedrock geology shows that the Britannia formation and the Britannia sills make a similar change in strike. The shear zone is almost entirely located in the porphyry of the Britannia sills and, as far as known, to the Fairview sill, the sill in which the Fairview ore-body is located.

"In the shear zone the quartz porphyry is changed to a fissile, greenish quartz-mica-schist, spotted with green films of chlorite up to an inch in length. This schist is locally called the Britannia schist. All gradations exist between the porphyry and the greenish quartz-mica-schist.

"*Mineralization.*— Mineralization is confined to the shear zone at irregular intervals. The ore-bodies of the Jane and Fairview occur where the shear zone shows a decided change in strike. As this was the only area of mineralization examined it cannot as yet be stated that all mineralization of importance occurs at such points, but it is worthy of consideration and study. In the mineralized area the ore-bodies occur as lens-

shaped masses separated by barren ground of various thicknesses. Mineralization is of the nature of impregnation and replacement of the schist by pyrite, chalcopyrite and cupriferous pyrite, with minor amounts of zinc blende. The gangue is almost entirely the quartz schist which in places is highly silicified."

Camsell, in his description of the Indian River section of the Coast range, says:

"Indian river is a stream about 20 miles in length flowing south into the north arm of Burrard inlet about 25 miles from the city of Vancouver. It occupies a steep-sided, heavily-wooded valley in the Coast mountains which rise in massive peaks, 3,000 feet to 4,000 feet on either side. Claims have been held for a number of years on the upper part of the stream and on the divide separating Indian river from Stamish river. Recently, however, a large number of additional claims have been staked farther down the valley and much interest has been shown in prospecting these claims. A trip of three days was made at the end of August from Squamish to the mouth of Indian river and the more important copper deposits were examined *en route*.

"A wagon road runs up the valley of Indian river from the north arm about 6 miles to an abandoned lumber camp and from that point a trail extends up to the head of the river, across the divide, and down the valley of Stamish river to Squamish. The trail on Stamish river is, however, not in good repair and some work would have to be done on it before pack horses could be taken through.

"A band of stratified rocks a mile or more in width, consisting of schists, andesites, argillites, and some limestone, extends more or less continuously in a northwesterly direction along the valleys of Indian and Stamish rivers. It lies a few miles east of the Britannia mineral zone and is separated from that zone by a belt of grano-diorite. The rocks are all highly metamorphosed by regional as well as contact igneous action and the limestone shows a peculiar silicification whereby kernels of quartz have been abundantly developed in it. Dips and strikes are difficult to determine, though the attitude of the beds is usually highly inclined and their strike conforms in

general to the trend of the whole band. They are cut and surrounded on all sides by grano-diorite of the Coast batholithic intrusion and are traversed by numerous dykes of aplite, quartz porphyry, granite porphyry, granodiorite porphyry, and other more basis dykes.

"The deposits all contain copper as the principal valuable metal and occur either as irregular replacements of the stratified rocks or in the dykes themselves after enrichment by mineralizing solutions. Chalcopyrite is the only copper-bearing mineral noted, but associated with it are much pyrite and some zinc blende. Most of the deposits are of somewhat different character to those of the Britannia mineral zone, though their origin may be attributed to the same general causes and the same period of metallization.

"Owing to the abundance of sulphide mineralization in the igneous rocks, and especially some of the dyke forms, and the widespread distribution of these dykes throughout the whole band of stratified rocks, the geological conditions are favourable for the occurrence of important bodies of copper ore, and it is reasonable to expect that such will be proved to exist when prospecting and development work have been carried farther. The difficulties presented to the active prospecting of this district are great, however, owing to the heavy forest growth and the deep covering of drift. The situation, too, of the mineralized zone, 12 to 15 miles from tidewater and served only by a rough pack trail, will make the progress of development slow."

## MINERAL CLAIMS IN THE COAST RANGE.

In the southerly portion of the Coast range, that is, between the Fraser river, on the south, and Seymour inlet, opposite the north end of Vancouver Island, a few prospectors have been diligently at work during recent years. The result of their efforts has been the discovery of a number of ore-bodies, such as that on the "Lucky Four" property in the Chean range just south of the Fraser river and about twenty miles east of the town of Chilliwack. In the "Lucky Four" group the country rock is very similar to that of the Britannia mine, but the ore is generally a higher grade chalcopyrite. Outside of the mineralized zone known as the Britannia belt, copper

sulphide ores have been discovered on the following groups of mineral claims north of the Fraser river: the "Attorney"; the "Opporgol", on the easterly side of Howe sound, near Whytecliffe; the "Horseshoe", on the westerly side of Howe sound, nearly opposite the Britannia mine; the "Bruce", "Contact", "McKinnon", and "Radiant", on Ray creek, a tributary of the Stawamus river, near Squamish; and the "Bowena", on Bowen island.

On Indian river, in the belt of stratified rocks referred to in Camsell's report quoted above, several groups of mineral claims have been located between the head of the river and a point about 10 miles below. The most prominent of these groups are the "Belle", "Bulliondale", "London", "Roy", and "Caledonia". Within the past few years all of these groups, except the "Belle" and "Bulliondale", have been acquired and crown-granted by the Britannia Mining Company.

Along the line of the Pacific Great Eastern railway, in the vicinity of Brew and Mona stations, there are several chalcopyrite deposits, the most prominent being the "Fitzsimmons", "Summit", "Green Tree", and "Alpha" groups. There is, also, a deposit of bog iron ore or limonite near the north end of Alta lake.

Travelling up the coast to Jarvis inlet, copper sulphide ore occurs near Egmont point; at the head of Narrows arm on the Zyoone river; and also near the head of Salmon arm, the last-named having been already referred to as the result of the first prospecting in that section.

On Treasure mountain, near Vancouver bay on Jarvis inlet, there are extensive deposits of low-grade copper sulphide ores associated with magnetite; while on the opposite side of Jarvis inlet, about six miles from the mouth of Britain river, outcroppings of high-grade chalcopyrite occur and several mineral claims, known as the "Lillie" group, have been located. At the head of Hotham sound, an arm of Jarvis inlet, on the easterly side near its entrance, there are several outcroppings of copper sulphide ores, and a considerable amount of development work has been done on some of these.

In the mountainous country between Powell lake and Theodosia arm bands of altered sedimentaries occur in which are found copper sulphide ores carrying low values in gold and

silver. On Goat island, in Powell lake, outcroppings of ore of the same character also occur, and similar ore has been found in the mountains adjacent to the Olsen river, as well as near the foot of Powell lake.

Between Powell river and Lund, deposits of ore, whose chief metal content is zinc, occur and have been developed to some extent.

The central and southwestern parts of that portion of the Coast range described in this paper are favoured with good conditions relative to travel and transportation, which are rendered comparatively easy owing to the several navigable lakes and rivers on which small boats and canoes can be used.

In the early history of the Cariboo country these favourable conditions were recognized and taken advantage of when the route was established from near the present site of Chilliwack, on the Fraser river, to Lillooet. The route follows the Harrison river to the foot of Harrison lake; up that lake for its entire length of 40 miles to the head, where Port Douglas was established; then by way of wagon road paralleling the Lillooet river to the lower end of Tenas lake. After canoeing to the head of this lake, a six-mile portage leads to the lower end of Lillooet lake, which is about 20 miles long with the ranchery of the Pemberton Indians located at its head. A wagon road connects this with the head of Anderson lake, the trip down which is made by steamer or canoe to a short wagon road between the foot of Anderson lake and the head of Seton lake, which is about 20 miles long and empties into Cayuse creek near the town of Lillooet. Lillooet, during the early Cariboo days, was a place of considerable importance, and placer mining was its chief industry until quite recently. The route just described enables one to reach the central portion of the Coast range very easily and without many of the discomforts and inconveniences which are inseparable from the prospector's life in the more northerly latitudes.

During 1897 and 1898 quite a lot of attention was directed towards that portion of the Coast range near the head of Harrison lake, as well as to the mountains adjacent to its shores. What was known as the "Fire Mountain Stampede" occurred in 1897, when, as the result of the discovery of high-grade gold-bearing quartz veins in that district, prospectors

rushed into the mountains on the north side of Fire lake, about 10 miles from the head of Harrison lake, and located over 50 mining claims in the course of two or three days.

One company that was organized to take over a group of mineral claims, known as the "Money-spinner" group, erected a Huntington mill and constructed a tramway to connect the mine works at the Money-spinner claim with the mill, nearly 2,000 feet distant. Unfortunately for the mining company and for the district, the milling plant was built before the ore-bodies had been properly developed, and after the plants were erected the company lacked the necessary funds to carry on development at the mine; consequently all operations ceased, and they have never been resumed.

It was a case of giving a dog a bad name and then hanging him.

In the vicinity of Agassiz, on the C.P.R. about 70 miles from Vancouver, there are several occurrences of copper ore, carrying bornite and chalcocite. Some of the claims that have been staked on these ores have been developed to some extent. Geologically, these occurrences are very similar to those described by Camsell and Schofield.

That no active operations are being carried on at present on any of the properties in the southerly part of the Coast range (except at the Britannia mine) is very much to be regretted; the owners of the properties ascribe this state of affairs to various causes, but generally speaking these may be summed up in the words "lack of capital".

On Fanny bay, Ramsay arm, there is a deposit of galena and zinc blende. In the vicinity of Frederick and Phillips arms, off Cardero channel, occur the gold-bearing quartz veins already referred to — the "Dorothy Morton," "Alexandra", and Blue Bell" — while near the head of Frederick arm, on the "Colossus" property, there is a deposit of copper sulphide ore. Northerly from the last mentioned occurrences, near the head of Knight inlet, there are bands of altered sedimentary rocks which have been intruded by the Coast Range granite, and in these sedimentaries are found ore-bodies carrying chiefly copper sulphide ores.

Between Knight inlet and Seymour inlet, the entrance to which is on the southerly side of Queen Charlotte sound, there has been but little prospecting up to the present time. At the head of Seymour inlet, deposits of magnetite of considerable extent occur.

Prospecting has been carried on during recent years in the vicinity of Tatlayoko lake and at the head of the Klina Klina river in the northerly part of that section of the Coast range dealt with in this paper. High grade gold-bearing quartz has been discovered in the first mentioned locality, and a very fine grade of red hematite in the last named.

Both these sections have been visited by myself and they appear to me to present very attractive possibilities as fields for systematic prospecting. The only handicap existing at present is lack of transportation facilities connecting these localities directly with the coast centres; consequently there is difficulty and extra expense incurred in transporting supplies. Eventually, after prospectors turn their attention more seriously towards the Coast range, this handicap will doubtless be removed.

The ore occurrences just mentioned occur on the easterly side of the Coast range in close proximity to the contact between the igneous rocks of the batholith and the older sedimentaries of the Interior Plateau. This section is practically unknown, except to a very few prospectors and to some trappers.

Bute inlet and the Homathka river cross-cut the entire Coast range from the coast to Tatlayoko lake, a total distance of about 130 miles; while Knight inlet and the Klina Klina river cross-cut it from the coast to the head of the Klina Klina, a total distance of about 130 miles; consequently, prospectors are afforded excellent opportunities to explore the mountains from central camps, or distributing centres, which they can establish along the shores of the inlets or the valleys of the rivers. To add to the attractive features of this portion of the Coast range, it might be mentioned that fish and game are fairly plentiful, especially such big game as mountain sheep, goats, bear and deer.

# GENERAL MINING AND MILLING PRACTICE AT THE BRITANNIA MINE, HOWE SOUND, B.C.

By C. P. Browning.

Annual General Meeting, British Columbian Division, February, 1922.

The property of the Britannia Mining and Smelting Co., Ltd., known as Britannia Beach, is situated 28 miles north of the city of Vancouver, on Howe Sound. The concentrating mill, bunkers, general offices, etc., being at tide water on the east shore of the Sound, a navigable arm of the Gulf of Georgia, are thus afforded convenient water transportation for supplies to the property and for concentrates from it to the Tacoma smelter. Three miles east of the Beach Camp, at an elevation of 2,100 feet, is the Mine Camp, from which point the bottom level of the mine is entered. The eastern limit of the property, which consists of about 560 mineral claims totalling some 25,000 acres, is ten miles from Britannia Beach.

This section of British Columbia is very rugged, and is heavily timbered with hemlock, larch, cedar, and red fir. The average annual precipitation is 80 inches, with a temperature ranging from 10 deg. to 85 deg. F. Only a few inches of snow ever fall at sea-level, but at an elevation of 4,300 feet, on top of the mine, it is not uncommon to have a depth of twenty-five feet. This snow, remaining in the hills until the summer months, gives a fairly uniform supply of water-power for operation during the entire year.

*Ore Deposits.*—In the Britannia area, mineralization is confined to a shear zone entirely within a belt of metamorphosed sedimentary and igneous rocks which form an inclusion in the grano-diorite batholith of the Coast Range. Five ore bodies in the same shear zone are included in the term Britannia Mines. Enumerating them from west to east they are: Jane, Bluff, Fairview, Empress, and Victoria.

The Jane deposit was the first to be worked on a commercial scale. This zone, irregular and roughly diamond-shaped, is extensively mineralized for a length east and west of 300 feet and over a width of 250 feet. The greatest depth at which commercial ore has been found is less than 300 feet below the surface. The ore in the Jane zone contains pyrite, zinc blende, chalcopyrite, galena, and possibly barite, with scattered gold and silver values.

Nine hundred feet north-east of the Jane ore body is the Bluff deposit, having a maximum length on the 1,200-ft. level of 500 feet east and west, and a width of 250 feet north and south. This body extends to a known depth of 1,600 feet below the surface. It is very massive, highly silicified, and extremely hard. The average dip of the whole body is 60 deg. to the south, with a general pitch of from 20 to 25 deg. to the west. The ore minerals are pyrite, chalcopyrite, zinc blende, galena, and argentite, with a gangue of quartz. Gold and silver values are persistent but lower than in the Jane body.

To the east of the Bluff deposit is a transition zone to the Fairview deposit. This latter is a much softer formation, with a maximum mineralized area of 1,500 feet in length and 500 feet in width. The occurrence has a known depth of 2,200 feet, with a strike of S. 80 deg. E. and an average dip of 72 deg. south. The schist is extensively mineralized with pyrite and massive chalcopyrite, in lodes varying in width from 10 to 70 feet. Between these lodes, locally called veins, there are zones of barren or very low-grade schist which also vary in width. The foot and hanging walls of the zone are not clearly defined, lack of mineralization in the schist giving the demarkation.

The Empress ore occurrence is about 1,000 feet east of the Fairview and on the west side of Furry creek valley. Its maximum length is 1,000 feet with a width of 100 feet. The general strike is S. 70 deg. E. with a varying dip, averaging 63 deg. south. Pyrite, chalcopyrite, and chalcocite are the ore minerals, and the surrounding rocks are soft and much weathered, Leaching has taken place in the upper portions of the veins and secondary enrichments occur.

East of the Empress ore-body is the so-called Victoria deposit, in which the schist zone is heavily mineralized with

pyrite and chalcopyrite. Neither the boundaries nor the dimensions of this ore-body have yet been determined, but no doubt it will prove to be of considerable size. It is badly broken and very much crushed in parts and is overlain by a depth of 150 feet of surface wash, making it difficult to prospect rapidly.

*Mining Methods.*—Most of the veins or lodes, standing as they do at an angle of approximately 72 deg., with a width of from 10 to 70 feet, and having fairly strong walls, lend themselves most economically to the overhand shrinkage stoping system. In most cases the ore in these stopes is drawn through a continuous line of chutes spaced 33 feet apart on the levels. These chutes are cut from the drift previously driven in the vein on its footwall side.

From the top level of the mine, a shrinkage system drawing to incline raises, in combination with the 'glory hole' method on the surface, has proved very satisfactory. This latter method of underhand stoping has been carried on during the last five years and can be employed for about seven months of each year; during the remaining months an accumulation of snow prevents this work on the surface.

When the mine was first opened square set stopes were the rule, but these were discontinued in 1912 because of the excessive cost of the method and because the ground was amenable to the less expensive shrinkage stope system. Undoubtedly some timber method will be required in the exploitation of the Victoria deposit.

*Mining Operation.*—A shrinkage stope is opened up by driving funnel-shaped chute raises, 33 feet centre to centre (three to 100 feet), from the foot-wall side of the drift, which has previously been driven along the vein on the level. These chute raises are advanced on an angle of 70 deg. with the dip of the formation to a distance of 20 feet above the sill before the timber chute is built. After the timber is in place, adjacent chute raises are connected at a point about 25 feet above the level, which leaves a strong wedge-shaped pillar of solid ground over the drift. When these chute raises are all connected in any particular section of the vein, the stope, as it is then called, advances upwards, leaving one chute at the end of the stope open to the level, for a time, as an entrance and exit from the stope.

Simultaneously with the cutting of the chute raises, a man-way raise to serve the stope is being driven from the same level to the level above. These man-way raises, 6 x 10 feet, are spaced about 300 feet apart on the drift, at right angles to the vein on its dip. On either side of the man-way raise, along the vein, a 10- to 15-foot pillar is left, so that this man-way is kept entirely surrounded by the rock pillar. At 30-foot intervals in this raise, by-passes are cut in the vein through the enclosing pillar, on two sides of it, to be used as entrances into the stope when the latter has been raised to that point. The solid roof or back in the stope is drilled by the miners while standing on the broken ore. After each blast the broken ore is lowered by drawing the chutes on the level below. When this ore has been drawn low enough for a repetition of the drilling operation, the miners bar down loose pieces and drill as before. As the stope proceeds upwards towards the level above, each successive by-pass into the man-way raise is used for entrance and exit for men and supplies as well as for the air pipe line.

Besides the chute pillars just above the drift level, and pillars around the man-way raises, other arch pillars are left, when needed, between the foot and hanging walls. These pillars vary in size but often have a base of 25 feet, and a length of 25 feet, with a sloping top so that the ore will not lodge on them when drawing. A sill pillar is left between the upper limit of the stope and the level above. This pillar varies in thickness from 20 to 30 feet, depending upon the width of the stope and the character of the ground. When all stopes on any specific vein are completed, part of these sill pillars may be taken out and recovered. The length of any one stope of the same name is the length of commercial ore on that vein between levels, but the stope is divided into independent sections by the man-ways mentioned above, spaced approximately 300 feet apart. The height of a stope varies with the distance between levels, which, to meet all requirements to the best advantage, is 150 feet. No timber is used in the stopes.

In a stope not exceeding 25 feet in width, the drilling of the solid back is as follows: A V-cut is drilled across the stope, then rows of seven-foot 'uppers', spaced three feet apart in average ground, are drilled to break to this cut. If the stope

is wider than 25 feet, a cut is taken only in the centre of the stope — not across the entire width — and holes are drilled around this cut to break to it as before. In all cases the cut holes are blasted first, and care is taken that they are out clean before the other holes are loaded. As far as possible the ground is drilled and blasted so that ore is thrown to the section of the stope directly over a point of drawing.

There are two mining shifts a day; the blasting being done during the third shift. The drillers do not blast in the underground stopes, as this part of the operation is performed by a specially trained crew of blasters who make the most economical use of the different grades of explosives. Effective shattering of the rock is desired as well as breaking to the bottom of the holes, thus eliminating as far as possible blockholing in the stope or breaking large boulders when drawn to the chutes. Good ventilation is maintained by natural draft through the man-ways.

The stoping drills used are the 14A Waugh stopers which, have proved very satisfactory under local conditions. These machines have been fitted with a water spray attachment for allaying the dust, but this has not been entirely satisfactory, the men preferring to take the dust rather than be bothered with the adjustments and choking of the spray attachment, or the chance of getting wet. A solid 1¼-inch cruciform steel of Rocky Mountain section is used. The ordinary crossbit has been found to be best for stoping work.

As mentioned, a shrinkage system drawing to incline raises is followed out at the top of the mine. From the uppermost level, about 200 feet below the surface, two 50-deg. untimbered raises, 6 x 10 feet, are driven from a common point. One of these is driven east and the other west along the strike of the vein. These are raised through to the surface, and may be used as man-ways from which an underground shrinkage stope is worked, with the two raises as the east and west limits. The operation of driving another pair of raises is repeated along the vein, and so on. Between these respective pairs of incline raises there would be driven one vertical raise, into which the ore left in the remaining pillar would be mined. During the summer months the outcomes of the raises are enlarged and the "glory hole" method is carried on. The

width of the vein determines the width of the "glory hole" and the length is determined by the raises. Arch pillars are left whenever necessary for steadying the ground.

A 9-foot vertical depth of hole drilled in "glory hole" work is found most economical. These holes are drilled with No. 60 Dreadnought pluggers, without water, but using air-blow valve, one man to a machine. Hollow round steel of 1¼ inch diameter is used. Blasting with electric caps and 100 shot battery gives good results, causing less delay at time of firing and better confinement in the "glory hole" of the round breaking, as the blast is more or less as a sheet mass, with ample shattering. The miners on this operation do their own blasting, whenever the round on any specific bench of ore is completed. Only one shift a day is maintained on this surface work. At the level on which the incline raises start, the ore is drawn off through the apex of the raises directly into long transfer raises below. Before going into these latter, the larger pieces of ore are bulldozed to pass between grizzly rails set twenty inches apart.

*Development.*—Drifting in average ground, a round of 18 holes with bottom cut is drilled. In hard ground the four-hole centre pyramid cut is used, and from 19 to 25 holes are drilled for an 8 x 8 foot section. The round varies with the number of relievers. Cut holes are blasted prior to the loading of the balance of the round and the advance per round averages 5½ feet. A standard type of round is not insisted upon, as the ground is not uniform, and the machineman, with the help of the shift boss, may take advantage of slips or seams in the placing of his holes. Cross-cuts are driven between veins, usually in a north-south direction, 8 x 8 feet in section. The operation of driving these is similar to that of drifting, but as a rule breaking better with fewer holes. Contracts are let for any drift of appreciable length, with tools, sharp steel, and air supplied to the contractor. He buys explosives and blasts, also handles the broken rock. Whenever possible a contractor is given two faces on which to drive, thus letting him have a chance to use his men to the best advantage in drilling and shovelling.

Steeply dipping raises are driven 6 x 10 feet in section, allowing room for ladders, pipe, and small skipway, besides

the rock compartment. A skip for handling steel and other supplies to the face is operated by a small reciprocating air hoist located near the bottom of the raise. Raises driven on a 50-deg. slope are untimbered, the broken rock being drawn out after blasting each round.

*No. 1 Shaft Equipment.*—The No. 1 Shaft, for the passage of both men and supplies between the 2,200 and 1,000 levels, is an underground, vertical, three-compartment shaft, 1,300 feet long. The hole cut was 10 x 20 feet in the clear, giving two compartments 6 ft. 0 in. x 7 ft. 6 in. and one 3 ft. 0 in. x 7 ft. 6 in. This shaft was raised from the bottom and is timbered throughout with 12 x 12 timbers, sets being 6-foot centres. Double deck cages are operated in balance by a 10-ton capacity double drum electric geared hoist. A 1¼-inch rope is used on 6 ft. 6 in. diameter grooved drums.

*Explosives.*—Two types of high explosives, gelatine and ammonia dynamite, are used, 40 per cent strength of the former, 30 per cent and 40 per cent of the latter. The gelatine dynamite is used in wet holes, and because of its plasticity it is better in breaking out 'boot-legs.' For confined work, such as in raises, it has the advantage of giving off less fumes. From quite extensive tests, however, the 40 per cent ammonia dynamite is found more economical for general blasting. Its detonation is faster than 40 per cent gelatine, so that it has a greater shattering effect; and, with the explosive strung out along the hole, the tendency is to break the ground into small pieces rather than large blocks. The 30 per cent powder is used in easy breaking ground. The ammonia dynamite is drier than gelatine, but when tamping is used there is no chance of the powder shaking out of any angle of hole drilled. Tamping facilitates the breaking effect in all cases. For contractors' use, the same types of powder are wrapped with red, instead of the usual light yellow, paper. This eliminates the chance of the contractor using company powder, for, even after blasting, small bits of the wrapper may be found through the rock piles as evidence of the powder used.

No. 7 strength detonators are used for all general blasting, with No. 6 strength for bulldozing. No. 6 10-foot electric detonators are used to some extent for the glory hole work, with rack bar blasting machines. The standard fuse is made

of countered gutta percha covered with a white water-proofing, and has a burning speed of 40 seconds per foot. One powderman cuts and crimps all the fuse, employing a mechanical cutter operated by hand, which cuts 30 fuse at one throw of the blade.

The main storage magazine for powder is located on a flat grade switch off the main line of the tunnel railway. It is situated 3,000 feet from any dwelling house and is amply isolated by an intervening rise of ground. The magazine has a capacity of 45 tons and is 20 x 20 feet inside, with cement mortar walls and trussed roof of the same material, on a one-third pitch, over-laid with corrugated iron roofing. Small well protected ventilators, front and back, give good air circulation, and a heavy cement door, 5 x 7 feet, slides on a substantial track. The floor of the magazine is level with the top of the flat cars.

*Steel and Bits.*—In drilling hard ground in the stopes, the 1¼-inch cruciform steel is used with a cross bit having two tapers, 5 deg. and 15 deg.—in easy drilling the 15-deg. only — both having a 100-deg. cutting edge. The length of starter is 2 ft. 6 in. with 2¼-inch gauge, and each successive steel increases one foot in length with a ⅛-inch decrease of gauge. The length of the 5-deg. taper on the starter is $\frac{3}{16}$ of an inch, which decreases to ¼-inch on the sixth steel, 7 ft. 6 in. long. For all drifting and cross-cut drilling, the 1¼-inch hollow round steel with the ordinary cross-bit is used. This bit, for hard rock, has a single 13 deg. taper, and for softer drilling a 15 deg., both having 100 deg. cutting edges. In block-holing with jack-hamers, the ⅞-inch hollow hexagon steel with cross bit is standard use. There has been a good deal of experimenting with Carr, double arc, H-bits, etc., but for local conditions the cross bit as mentioned gives the most satisfactory results, with much greater ease in forming. Lengths of drifting steel run from 2 ft. 6 in. to 10 ft. 6 in. with a one-foot change, the guage decreasing ⅛-inch at a time, from 2⅝ inches on the starter to 1⅝ inches on the ninth steel.

For sharpening the steel there are nine shops throughout the mine, advantageously located at portals close to working stopes. Ventilation pipe from a furnace to a portal is necessary

in only one case. In seven of the mine shops oil is used as fuel, but in the other two,because of difficulties in the handling of oil barrels to them, the fuel is coke. The fuel oil is 18 deg. B., and flows by gravity to the furnaces after preliminary heating in the coils around the furnace stack. One man and helper will sharpen and temper about 450 steel per 8-hour shift. The sharpeners in use include No. 5 Leyners, Waugh, and Sullivan machines. Close to each sharpener is a mechanically operated punch for clearing and enlarging the hole in hollow steel. Near the bottom of one of the main working shafts is a 20 x 40 foot underground shop, equipped with a drill press, emery wheel, air meter, machine testing block, and pipe threading machine, for the repairing of machine drills. This obviates taking the machines to the surface for repair or testing. Record of all repairs and replacements is kept at this point.

The mine shop for general repair work on cars, switches, small hoists, machine parts, etc., is located at the portal of the 2,200 level. The equipment consists of emery wheels, a No. 800 24-inch stroke air operated hammer, a drill sharpener, shaper, a milling machine, drill press, two lathes, power hack saws, and a bolt threading machine. A few notes on shop practice may be of interest: Dies and dollies for the machine sharpeners are forged and machined at this outside shop. Worn-out mill roll shells and locomotive tires give excellent material for dies and fuller blocks, but it is found too brittle for use in dollies. Clamping dies are forged in halves, faces machined, then sweated together with solder, drilled and finished. They are well annealed in lime and tempered in lard oil. The shanks of the machine steel are camphored and then tempered in lard oil. For shanking 1¼-inch hollow round steel from the bar, three heats are necessary. The heat is applied just beyond the lugs, and the hole in the shank is enlarged to ½-inch diameter. The ½-inch diameter pins used with dollies for hollow steel are inserted 1½-inches, and from that point through the rest of the dolly is drilled a $\frac{5}{16}$-inch hole, making it a simple operation to remove the dolly pin for replacement or repair. Dolly springs are made of wires from short pieces of hoisting cable. Nine .08 m. diameter wires are twisted together in the lathe and then bent to suit the sharpener. Oil bottles for the individual miner's use underground

are made of a 14-inch length of 1½-inch pipe, closed at one end and welded to a short ¾-inch nipple at the other. This nipple, with a screw cap, forms the neck and stopper of the bottle. The whole forms a strong, handy, tight bottle, and one that is not carried away by a miner when quitting. The drill sharpener located at the mine shop is equipped to hammer out track spikes, bolts, and ship spikes from any scrap rods.

*Tramming.*—Ordinary swing-dump cars of 26 cu. ft. (1½-ton) capacity are used for hand tramming. These cars are fitted with 12-inch solid, chilled iron wheels, oiled through a spring cap on the outside of the hub. A 24-inch gauge is standard throughout the mine, and for hand tramming 12-lb. rails are used with a maximum grade of ¾ per cent with the load. In drawing ore from the chutes, pieces too large for the cars are block-holed and blasted.

The cars used with the storage battery locomotives are 33 cu. ft. (2-ton) capacity, automatic side dump, with cast steel spring draw heads, reinforced door, and six-inch diameter dumping wheel. They are equipped with 16-inch roller bearing cast iron wheels, with a wheel base of 2 ft. 11 in. The car is 4 ft. 2 in. in height and operates on a 20-lb. rail. The cars dump automatically without stopping, as the dump wheel engages with an incline bar opposite the transfer raise. Storage battery locomotives, both 3½- and 6-ton weights, are used on all the main levels, and will operate on a curve of 20 foot minimum radius. The batteries are of the iron-clad excide cell type, and the motors have a tractive effort of 2,700 to 3,000 pounds. The underground charging stations are located close to the shafts on their respective levels, while motor generator sets for supplying the D.C. power have been installed at the hoist stations. They are thus operated by the hoistmen, who are entirely familiar with this class of machine. The charging and running of these motors conforms nicely to the operation of two mining shifts per day. On one of the main working levels, three trolley locomotives are running, of which two are 3-ton and the other a 6-ton. A 250-volt D.C. circuit supplies the power for the locomotives, which operate on a 30-lb. track, handling 2-ton cars.

*Transfer Raises.*—As far as possible, the ore is handled by gravity through long transfer raises centrally located in ore on the dip of the vein. All ore is trammed with the least possible handling to the main ore raise, which delivers it directly to the crushers on the 1,700 and 1,800 levels. Ore may be dumped into this main ore raise on all the levels from the 1,800 up to the 700 level. The individual by-passes, 5 x 9 feet, are cut to the hanging wall of the raise on a 60 deg. slope, then enlarged at the level to accommodate the dumping of the 2-ton cars. Over the collar of these by-passes is constructed a grizzly with rails set 20 inches apart, the ends of which are held on 12 x 12 timbers, doubled in some cases. The grizzly rails are made up of two pieces of heavy rail, reinforced with three pieces of 1 x 3 inch iron, all bolted together. None of the raises are timbered and the falling ore not only breaks itself up, but in time enlarges the raises as well. The storage capacity of transfer raises, exclusive of the main one, is approximately 25,000 tons. The main one holds 7,000 tons above the crushers and 2,500 tons between them and the 2,200 level, where the ore is drawn.

*Preliminary Crushing Underground.*—The run-of-mine ore is given a preliminary crushing to a 6-inch size at the 1,700 level in a 24 x 36 inch jaw crusher. It is then delivered by gravity to either one of two 7½ K. gyratory crushers on the 1,800 level, which reduce the ore from 6-inch to 2½-inch ring. The crushers are all preceded by sizing grizzlies, and each belt is driven by 75-H.P. motors. After the above, the crushed ore discharges into the main ore raise, dropping to the 2,200 level, at which point it is drawn into 17-ton cars, operating on the tunnel railway.

*Ore Transportation.*—Until December, 1920, at which time milling was stopped, due to economic conditions, the preliminary crushed ore was handled by two systems of transportation to the concentrator at the Beach. One was a combination of a 1½-mile three per cent railway grade and a 2½-mile aerial tramway, while the other was the combination of the 3½-mile three per cent grade and a 6,000-foot thirty per cent incline railway. Since that time the latter system of ore handling has been supplanted by a combination of the 3½-mile three

per cent railway and a 1,500-foot 65 deg. rock raise, which delivers the ore by gravity to a ¾-mile half per cent underground grade that takes it to the mill bins.

*Milling.*—As mentioned above, milling operations were stopped on the first of December, 1920, and in the following March the entire concentrating plant was destroyed by fire. This so-called No. 2 mill was of timber construction, having a good side-hill location where it was possible to take advantage of gravity flow in the milling. The ore, after passing over picking-belts, was crushed by 10 x 20 inch jaw crushers; then by 48 x 16 inch spring rolls; and next by 40 x 15 inch spring rolls. After this reduction and subsequent screening, the undersize product went to the Hancock jigs, the oversize going directly to 36 x 15 inch crushing rolls, which were followed by a battery of 36 x 15 inch fine crushing rolls. The product from these latter was then delivered at 10 mesh to sixteen 7 x 12 foot tube mills, which crushed the ore for oil flotation feed. Six 14-cell flotation machines were in use, which produced a 14 per cent concentrate, that was allowed to dry in the concentrate bins, thence being shipped by boat to the smelter at Tacoma. Overflow from the concentrate bins was laundered to the Dorr thickener tanks before going to the general tailings.

In the No. 3 mill, a steel and concrete structure of side-hill location, which is now under construction, there have been incorporated recent improvements in concentrating practice, so that the flow sheet of this mill differs somewhat from the former. There is a collecting conveyor under the coarse ore bins, delivering the ore, after passing over a vibrating screen and electric magnet, to one set of 72 x 20-inch spring rolls. The product from these rolls goes to four sets of 54 x 20 inch spring rolls, which reduce it to ¼-inch size. This is delivered to steel storage bins of 4,500 tons capacity. As the ¼-inch product is drawn from these bins on belt conveyors, it passes over weightometers and vibrating screens, the undersize going directly to six Hancock jigs and the oversize to eighteen 7 x 10 foot tube mills operating in closed circuit with classifiers. It is proposed to deliver this flotation feed from the tube mills to six 14-cell machines, the concentrates from which will go to three 50-foot diameter Dorr thickener tanks. The product

from these tanks will be dried in continuous filters, then shipped as concentrates to the smelter. The roll crushing section of the plant may be operated 16 hours per day for the 24-hour operation of the other departments.

*Power.*—The electric power is produced from water on the property. A series of storage dams hold a supply for the dry season, so that, by having one power house at an elevation of 2,100 feet and the other at the Beach, some of the water is made to serve for power twice before it reaches sea-level. Electric power is transmitted at 6,600 volts. A 3,000-K.W. steam plant is located at the Beach as an auxiliary power plant.

Thus, having the power entirely produced on the property, the whole operation of mining, transporting, and concentrating some of 2,300 tons of ore daily is more or less a self-contained unit.

# THE DOLLY VARDEN MINE.

By George Hanson.

Annual General Meeting, British Columbian Division, Vancouver, February, 1922.

The Dolly Varden mine, controlled by the Taylor Engineering Company, has been for several years the outstanding property in the Alice Arm section of the Nass River mining division, British Columbia. More than 1,300,000 ounces of silver have been extracted from its ores and some of the ore shipped has yielded over 1,200 ounces to the ton.

The mine is located at an elevation of 1,700 feet above sea level in the Upper Kitsault River district, 16 miles north of the town of Alice Arm. A narrow-gauge railway, 17 miles long, connects Camp 8 with the wharf at Alice Arm. The district can be reached on boats of the Union Steamship Company, which call at Alice Arm. Grand Trunk boats call at Anyox. Alice Arm is 18 miles from Anyox, and a tri-weekly boat service is maintained between the two towns.

## General Geology.

The Upper Kitsault River district lies a few miles east of the eastern margin of the Coast Range batholith, and the surrounding area is characterized by the rugged peaks and the steep sided valleys typical of Coast Range topography. The rocks of the area in general are folded into anticlines and synclines, which have a trend parallel to the eastern border of the batholith. Locally, folding transverse to these predominant structural lines has taken place.

The rocks can be classified into four main groups. Given in order of decreasing age, these are:

(1) A thick series of volcanic fragmental rocks;
(2) A series of argillites and other sedimentary rocks;
(3) Coast Range granitic rocks;
(4) Lamprophyre and diabase dyke rocks.

The oldest rocks exposed, the volcanic fragmental series of Triassic or Jurassic age, make up the country rock of the Dolly Varden mine, and of all the other silver properties in the district. The series includes some coarse agglomerates, but medium grained breccias predominate. The upper beds consist chiefly of finer grained tuffs. In general the rocks are green or grey in colour, but some members are purple or red. The breccias at the Dolly Varden mine dip in a northerly direction, and may be part of the northern limb of a small anticline transverse to the general structure.

Basic dykes are found in the mine workings, but as they do not appear to be related genetically to ore deposition they will not be discussed here.

Faulting has an important bearing on the ores. The faults are chiefly reverse faults and later than the primary mineralization. They strike roughly north and south, and dip 45-75 deg. to the west. The thrust has acted from west to east. These reverse faults are cut by normal faults having the same general strike and dip. The trace of a fault plane on the surface is marked usually by a narrow gully, but since the rocks are in general void of apparent structures, surface examination gives very little evidence regarding the nature of the fault, except where a vein or a dyke has been offset. Underground, there are other evidences, such as gouge, drag ore, striations, etc., which can be studied to throw light on the problem of determining the direction of movement of the fault blocks.

## DISCOVERY AND DEVELOPMENT OF THE MINE.

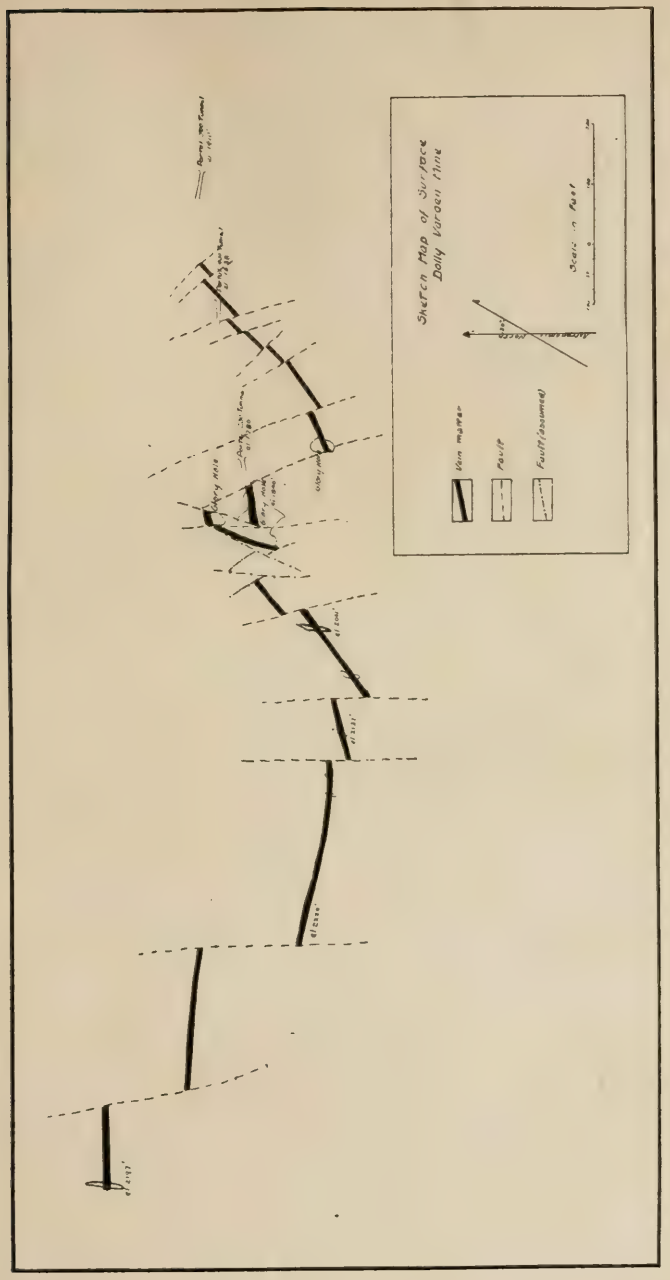
A prospector named Gus Pearson dreamed a dream. In his dream he found a mineral vein on a steep wooded hillside. The place was marked by a great white boulder. He staked the claim and named it the Dolly Varden, and eventually it became a big mine. Thus the dream. When Pearson and his partners were going up the Kitsault River valley later, they found the steep wooded hillside, found the big quartz boulder, staked a claim, and named it the Dolly Varden. This was the first mineral claim located on the upper waters of the Kitsault river.

The property was first described in the Annual Report of the Minister of Mines (B.C.) for 1913. There were then two open cuts showing a great deal of quartz, but carrying low values. The Dolly Varden was again mentioned in the Annual Report for 1915, and from that time to the present short descriptions have been given every year. In 1915 there were six claims in the group, and the property was under bond to R. B. McGinnis. By the end of the year a good deal of development work had been done, such as surface work, 550 feet of tunnelling, and 4,500 feet of diamond drilling. In 1916, work consisted of 800 feet of tunnelling and 4,000 feet of diamond drilling. A compressor was installed and a wharf built at Alice Arm. There were now eight claims in the group. Development, preparatory to shipping ore, was continued in 1917 and 1918. In 1919 the property came into the hands of the Taylor Engineering Company. In the same year, the railway was completed from Alice Arm to Camp 8. An aerial tram 1,900 feet long was also built from the 400-ft. level to Camp 8. To-day there are about 7,000 feet of tunnels, raises, drifts, etc. Production has been as follows:

| Year | Ore, tons, | Total Silver, oz. | Silver, oz. per ton |
|---|---|---|---|
| 1919 | 6,668 | 376,562 | 56 |
| | 42 | 50,562 | 1,202 |
| 1920 | 27,944 | 749,340 | 27 |
| | 93 | 82,298 | 884 |
| 1921 | 1,862 | 45,647 | 24 |
| Total: | 36,609 | 1,304,409 | 35⅔ |

## THE ORE BODY.

The ore body is in the form of a vein, of which only one wall, the hanging, is free. The footwall is gradational, ore merging gradually into country rock thoroughly impregnated with pyrite. In every place where ore has been extracted, the hanging wall consists of purple breccia and the footwall of green or grey breccia. The ore zone does not seem to be a pre-mineral fault, and consequently the vein is probably a bed vein. The ore material has formed along this contact, chiefly by replacement of the footwall rock. The hanging wall shows no replacement whatever. The vein east of the most easterly stope shows on the surface the same geological relations. It has not been developed underground. The westward extension of the vein is not a bed vein, but lies entirely in green breccia. The




Sketch Map of Surface, Dolly Varden Mine.

most westerly open cut, west of which the vein has not been traced, shows again the same geological condition as that of the stopes.

The vein has an east-west trend, and has been cut by transverse reverse faults into a number of sections, each about 100 feet long. The horizontal offset of the vein is more than 120 feet in some instances. (See accompanying plan.)

The ore is massive. Banding, crustification, etc., are absent. The vein material is highly silicious, with an average of 70% or more of quartz. Calcite and barite are other gangue minerals. The primary metallic minerals are pyrite, with a little chalcopyrite, sphalerite, galena, and tetrahedrite. Sphalerite and galena are more prominent than chalcopyrite, yet the combined percentage of lead and zinc in the ore shipped has been less than 2%. The primary vein, then, is really a silver bearing quartz-pyrite vein. The metal extracted is silver, and it has been taken from an ore composed chiefly of the minerals ruby silver, argentite, native silver, pyrite, and quartz.

## MINERALOGY OF THE ORE.

The metallic minerals present in the vein are pyrite, chalcopyrite, sphalerite, galena, tetrahedrite, argentite, pyrargyrite, proustite, pearceite, native silver, and limonite. The gangue minerals are quartz, calcite, barite, and jasper. Arsenopyrite[1], hornsilver and brittle silver[2], rhodochrosite or hodonite[3], have been recorded but were not identified by the present writer.

Pyrite is more plentiful than all the other primary metallic minerals together. It occurs, in the form of cubes, both in the footwall rock and in the vein itself, though in the vein-quartz it is commonly present as small stringers. The other primary sulphides are rare.

In the ore mined, native silver, argentite, and ruby silver were plentiful. Argentite is in part associated with galena, but, chiefly, it is present with the minerals which are distinctly

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[1] Forbes, Donald G., "Kitsaulte Copper Camp, Alice Arm, Observatory Inlet". Ann. Rept. Minister of Mines, B.C., p. K83, 1913.

[2] McMullin, J. H., "Skeena and Bella Coola Mining Divisions", Ann. Rept. Minister of Mines, B.C., p. K70, 1915.

[3] Turnbull, J.M., "Alice Arm District", Ann. Rept. Minister of Mines, B.C., p. K75, 1916.

later in time of deposition. Probably more silver was yielded by the native metal in the ore than by any other mineral. The native silver is present in the form of leaves and plates in small fractures in the vein matter; in the lower levels it is usually in wire form. With it are associated ruby silver, argentite and pearceite. The ruby silver minerals, particularly pyrargyrite, were important constituents in the ore. Limonite is found in the faults down to the lowest levels.

Quartz is by far the most abundant gangue mineral. It is milky white usually, but in much of the high-grade ore it is bluish to almost black. Barite is common in some parts of the vein, but is not associated with high silver values. These are in quartz gangue, here as elsewhere in the same district. Calcite in small quantities is found everywhere, associated with the other gangue minerals. It is more abundant in the wall rocks, as stringers of late deposition. Jasper, common in other veins in the district, is not plentiful in the Dolly Varden vein. The jasper contains a small amount of manganese.

The hanging wall rock has not been altered by ore solutions, but the footwall has been impregnated with pyrite. Some silicification and sericitisation has also taken place.

The group of minerals comprising pyrite, chalcopyrite, sphalerite, galena, and tetrahedrite, were deposited early in the process of vein formation. They are believed to be entirely primary. The ruby silver, argentite, pearceite, native silver group is of late formation, and of these, native silver was the last to deposit. The minerals of this second group are believed to belong to a second period of mineralization: that is, a period of secondary enrichment.

## SECONDARY ENRICHMENT.

The terms "primary" and "secondary" have been used with different meanings by various writers on ore deposition. However, nearly all writers now use the term "primary" to describe ore bodies whose chemical and mineralogical composition has not been changed by superficial agencies, and the term "secondary" to designate those ore bodies which have resulted through the action of superficial agencies on primary ore.

When mineral deposits are exposed to weathering, the compounds which are unstable under these conditions are

decomposed, and soluble salts and stable secondary minerals are formed. The soluble salts may move downward, react with minerals below, and thus form other secondary minerals. In the change from oxidizing conditions at or near the surface to what are, relatively, reducing conditions below, solutions may deposit part or all of their metal content. Secondary minerals may result from other reactions. In general the process of secondary enrichment takes place in three steps.

(1) Decomposition of primary minerals;
(2) Transportation, usually in solution;
(3) Deposition in concentrated form.

It is believed that the ore of the Dolly Varden mine represents enrichment of lower grade primary vein material. The evidence favouring this view will be presented under three headings:

(1) Geological evidence
(2) Mineralogical evidence
(3) Textural evidence.

*Geological evidence.*— If a high-grade ore is related in position to the present surface, it is probably related thereto in origin also. Again, if there has been considerable erosion since the deposition of the primary ore, the ores now confined to the surface are probably not primary but secondary. This point was brought out by Ransome[1] in 1910.

The primary ore bodies in the neighbourhood of Observatory inlet, as well as the great majority of all the primary metallic deposits of British Columbia, were formed in late Jurassic or early Cretaceous time. Since that time, erosion has stripped off thousands of feet of rock. Consequently, it would be a curious coincidence if the ore deposited at that time should show any close relation to the present surface. At the Dolly Varden mine, good ore has been mined to a depth of 200 feet below the outcrop, and development has extended to a greater depth. Mining and development have shown a great change from high-grade to lower grade vein matter between 175 and 250 feet below the surface. This is a strong argument favouring the view that the ore is secondary.

[1] Ransome, F. L., "Criteria of Downward Sulphide Enrichment"; Econ. Geol., Vol. 5, pp. 205-220, 1910.

Further evidence is obtained from the relation between the high-grade ore and the numerous faults. The best ore was found near the faults, and near the hanging wall of the vein. Since the faults are later than primary mineralization, a second period of mineralization is postulated to explain the relation. The faults and fractures are water-courses, and the obvious source of the mineralization of this period was the primary vein. A second period of primary mineralization is unlikely. There are dykes cutting the ore and one of them is later than the faults; but the dykes do not appear to have any relation to the ore. If the period were primary, the whole fault plane should be mineralized. This is not the case. Only the original vein carries ore. It seems obvious that the faults and fractures were simply channels for the downward-moving enriching solutions. The hanging wall is free and in many places has a slab-like or sheeted structure parallel to the vein. Solutions could also move downward along this wall.

*Mineralogical Evidence.*— In the faults previously mentioned limonite is ever present. This proves that oxidizing solutions have moved down along the fractures. The ore minerals regarded as secondary are argentite, ruby silver, pearceite, and native silver. The secondary nature of these minerals has been investigated by various writers. Emmons states that, in sulphide deposits, native silver is generally or invariably secondary[1], and that argentite is one of the commonest secondary minerals, but that it is sometimes primary[2]. He also states that the ruby silvers are nearly everywhere secondary minerals, and that pearceite is secondary in most of its occurrences[3]. Some secondary action is necessary to produce the limonite. It is probable that the other common secondary minerals were deposited at the same time and by the same processes.

*Textural evidence.*— In the upper workings of the mine, native silver occurred as thick plates and leaves. Lower down,

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[1] Emmons, W. H., "The Enrichment of Ore Deposits": U.S. Geol Survey, Bull. 625, p. 270, 1917.

[2] Emmons, W. H., idem. p. 274.

[3] Emmons, W. H., idem. p. 261.

the plates were thinner, and 200 feet below the surface very thin leaves and wires of the metal are found. All the native metal is found in small fractures in the primary vein material. In the lower workings, argentite and ruby silver are found only in the small fractures, but near the surface these minerals were more abundant, and occurred as well in large bunches and numerous small blebs. Near the surface, replacement has been more extensive than below and has gone beyond the fractures; in depth, however, replacement is confined to the fractures. It is not known whether all the fractures in which the secondary minerals are now found were previously occupied by primary sulphides. It is not likely that they were, however, as intermediate stages of this replacement are not common.

From a metallographic study of the ore, it is apparent that all the minerals believed to be secondary are later in time of deposition than the known primary minerals. It is also seen that argentite replaces galena in many places. Argentite areas show cores and remnants of galena. The argentite areas are in turn replaced around the edges by native silver.

## CONCLUSIONS.

Although it is fully realized that general conclusions, such as are given below, may be misleading if taken as rules, particularly since they are drawn in part from only one mine, they are stated because even one mine may contribute a good deal toward the "law of the camp". The conclusions given here refer to the Kitsault River district:

(1) The Dolly Varden contact between red and green breccia is a favourable horizon for ore.

(2) Red breccia is an unfavourable country rock.

(3) Green or grey breccia is favourable country rock.

(4) The primary vein contains primary ore shoots.

(5) Very high-grade secondary silver ore cannot be expected to continue in depth for more than 250 feet below the outcrop.

(6) More extensive faulting and fracturing is accompanied by more intensive enrichment, in veins of equal primary value.

## DISCUSSION.

MR. J. D. MACKENZIE: Two years ago last November there was a discussion about the Dolly Varden and other mines in the same district, and Dr. Dolmage was a protagonist at that time for the primary character of these ores. There is to some extent a tendency among geologists at the present time to believe that sulph-antimonides are more apt to be primary than secondary. Two years ago at this convention the tendency seemed to be to consider these minerals primary. I fail to see the justification of any interpretation other than that which has been put forward by Dr. Hanson, after a field study made in the mine. Materials were gathered on the ground, and, from these, facts, conclusions, and theories, are derived. Two years ago the discussion was based on small samples examined under the microscope. To-day we have the facts obtained in the examination of the mine itself, and there is no doubt Dr. Hanson has given us the correct solution. An important fact is the reduction in the grade of the ore. Finding the enrichment along the faults leads one to believe the enrichment has been connected with the surface water. Again, you have the limonite. These facts point to one conclusion, and one only: that is, that these ores are secondary.

MR. H. R. VAN WAGENEN: I am in accord with the views which have been expressed concerning the secondary enrichment of silver veins. So common is this action, because of the ease with which silver is dissolved by surface waters, that, in countries where the rate of erosion has not kept pace with the migration of the soluble metals, secondary minerals are invariably present. Small veins, of uncommercial grade where primary, have thus been made profitable near the surface. So high-grade are they, frequently, as to be a "poorman's proposition" and, as a result, in the arid "Silver States", such as Nevada, the name "chlorider" has long been the title given to the prospector or leaser who makes his living (and occasionally a "stake") from coyoting outcrops and shallow workings in which the chief valuable constituent is silver chloride. Even in a country like British Columbia, where erosion has been very rapid, secondary silver is to be expected; but it would be unreasonable to hope that the zone of that secondary en-

richment would extend deep — because, as stated, erosion has been rapid, rapid enough to have almost kept pace with the migration of silver.

DR. HANSON: In the Dolly Varden there is no gold. Although gold is present in neighbouring districts, there is very little in the Alice Arm district.

MR. F. W. GUERNSEY: Is there any evidence of a change of the water table at the Dolly Varden? I understand they get secondary minerals below the present water table. A good criterion we used to have was, that when the permanent water level was reached, we looked for any secondary enrichment to stop, or at least only to extend a comparatively slight distance below such level. It was always a question, however, where the original top of the water table was. I remember the case of a mine in Nevada, where one of our men reported that the shaft, although the property had not been worked for some time, was dry at the 500-foot level. A year or so afterwards he returned to make a further inspection, and found the water standing at the 400-foot level. Between the times of his visits an earthquake had occurred in the vicinity and evidently had changed the level of the water table over the whole district. The point I wish enlightenment on is, are these secondary minerals, mentioned by Dr. Hanson, deposited at any great depth below the present water level? Or is there any evidence of a change of this level?

DR. HANSON: In the Dolly Varden, the enrichment probably took place in Tertiary time, and you could not tell where the water table was at that time.

DR. S. J. SCHOFIELD: I am very much interested in the evidence of secondary enrichment of silver. There are geologists who hold that the native silver of the Dolly Varden is primary. Because they find evidence that some silver is primary, they conclude all silver is primary. Every case has to be decided on its own merits, and every mine studied by itself. In the Dolly Varden, the evidence is conclusive, and in the Premier it is conclusive. The manager of the Premier considers the high-grade silver values are secondary. There are high values in gold which will make the mine produce for

a long time. It looks as if the gold values are primary, and if so they will continue with depth. It is nice to have these high values in an ore-body, because they pay for the preliminary development work. If you have these enriched values it is easy to open them up, and of course this helps a lot in the development of the country. The ore-bodies occur at the contact between quartz-porphyry and tuff. In the Dolly Varden they occur at the contact of the purple and green tuff. If you examine the geology of the country you will see the greater portion is underlain by tuffs, and this enlarges the possibility for the occurrence of these high-grade silver ores.

MR. H. G. NICHOLS: I would like to ask Dr. Hanson if he can give us some further information regarding the structural conditions, especially the localization, intensity, and persistence of the fracture system. I raise the point in connection with the depth to which secondarily enriched ore may be expected to occur. Dr. Hanson has suggested a depth of 250 feet for the horizon of secondary enrichment, but if the process is to be associated with the function of the fracturing as affording channels for the downward passage of the mineral solutions, a knowledge of the conditions of fracturing is essential to an appreciation of the ore occurrence. I came across an interesting case recently in Mexico, where oxidation and secondary enrichment had been carried to varying depths in different sections of an ore-body as a result of a system of vertical fractures. A body of secondarily enriched ore was followed on one level to a point where a vertical line of demarcation between sulphide and oxidized ore was passed. Beyond this point, the ore was found at a horizon about 200 feet lower, for a certain distance, and then, an area of a more intense phase of the fracturing being entered, it dropped out of sight altogether, so far as development had gone when I was there. It occurs to me that similar fracturing conditions in the Dolly Varden area probably have a pertinent bearing upon the case, and to a similar extent.

DR. HANSON: The secondary enrichment gradually decreases to a limit of about 250 feet, but the fracturing goes further down. It contains no secondary enrichment, however, but there is limonite in the fractures below the enriched zone.

DR. W. L. UGLOW: I did not hear the first part of Dr. Hanson's paper, but I would like to ask if he has any evidence to present to show us whether there is a secondary sulphide zone in that northern portion of British Columbia? Why has it not been removed, as it has been in other parts?

DR. HANSON: In studying the Dolly Varden mine I came to the conclusion the ores are secondary. Whether or not these secondary ores should have been eroded, they were not eroded. The country has been glaciated, it is true, but in the valleys there is not the same evidence of erosion as you would expect if the ice had eroded deeply.

DR. UGLOW: Is the oxidized zone gone?

DR. HANSON: Yes.

# THE PREMIER MINE.[1]

By George Hanson.

Annual Western Meeting, Vancouver, November, 1922.

The Premier mine has been the subject of so much comment in mining circles that the writer felt that a short paper dealing with the geology and ore-bodies of the mine would be of interest. The statements which follow are of a general nature merely, being based on the results of a preliminary investigation.

## Location.

To reach the mine from Vancouver the journey is made by Grand Trunk steamer to the boat landing between Stewart, B.C., and Hyder, Alaska, — the landing is one and a half miles from the former town and one half mile from the latter. An automobile road extends from Stewart through Hyder to the mine, a distance of 17 miles. The road follows the Salmon river in Alaska to Eleven Mile, then ascends along Cascade creek for two miles, and finally zigzags up to an elevation of 2,000 feet above sea level on Bear River ridge. The road enters British Columbia 12½ miles from Hyder. The International Boundary at this point practically coincides with the coast Range batholith. The complex of pre-batholithic rocks extending to the north and south along the eastern border of the batholith is the great mineral belt of British Columbia.

## Geological History.

On a surface which is not now exposed, there was accumulated, in the latter part of the Jurassic period, a thick series of volcanic fragmental rocks of intermediate chemical compo-

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sition. The first half of this series to be laid down consisted of coarse breccias of a purplish colour, and these were followed by a series consisting of finer grained andesitic tuffs, which are either green or grey. Since the base of the series is not exposed, its total thickness cannot be definitely given, but it is greater than 2,000 feet. After the cessation of explosive vulcanism, ordinary aqueous sediments were laid down on top of the volcanics. These sedimentary rocks have a thickness of over 1,000 feet and consist mainly of black argillites. The rocks were then invaded by intrusions of quartz or feldspar porphyry, in the form of lenticular sheets or sills, which are present chiefly, if not exclusively, in the upper part of the series of volcanic rocks. For the most part, the porphyry magma forced its way along the bedding planes of the tuffs, but occasionally it cut across the bedding and thus surrounded great slabs of tuff. Quantitatively, the porphyry is not very important, but in its relation to ore deposition it appears to have been of the greatest significance. A good deal of pyrite is disseminated through the porphyry and it is possible that much of this pyrite had its source in the porphyry magma. There are numerous small gash veins of quartz, with a little pyrite and occasionally a little ruby silver, which lie exclusively in the porphyry, and it may well be that some of this matter was derived from the porphyry magma. It appears, however, that no commercial ore bodies were formed at this time. The date of the porphyry intrusion is not established. The rocks may have been gently folded prior to the intrusion.

During the closing stages of the Jurassic period many things were happening in this area and the sequence of events is not easy to ascertain. Folding of the rock formations continued, and the magma which eventually resulted in the Coast Range batholith was working its way upward underneath and to the west of the present site of the Premier mine. While being folded the rocks were sheared, faulted, and fractured. During these movements the tuffs were mashed and sheared rather than fractured, whereas the bands of porphyry were fractured more than sheared. The shearing and fracturing followed approximately along the strike and dip of the rocks. The Coast Range magma was solidifying, especially along the borders, and since the grano-diorite batholith is sheared locally it is evident that orogenic movements had not yet

entirely ceased. Later, granitic dykes, whose material was supplied from the still molten interior of the batholith, penetrated the bordering rocks, and since these dykes are not sheared or folded it is inferred that folding of the rocks had ceased at the time of their intrusion. Finally, basic injections proceeded from the interior of the batholith, and these have given rise to dykes which cut all the other rocks.

It was during the period of dyke intrusion that silicious ore solutions were coming out from the batholith and depositing their loads in the surrounding rocks. At this time, also, the first main period of fracturing was drawing to a close. The ore solutions traversed the fractured porphyry rather than the sheared and relatively impermeable tuffaceous rocks. The porphyry, moreover, appears to have been more susceptible chemically than the other rocks to the attack of the solutions. As a result, not only was the ore deposited in the fractures, but the solutions worked their way outward from these, partially replacing the wall rock for distances up to thirty feet. Occasionally the ore-bodies traverse areas of tuff included in the porphyry, and here the ore zone is usually much narrower than it is in the porphyry. Thus we find porphyry wall rocks silicified to distances of from one to thirty feet from the veins, while tuffacious wall rocks exhibit very little silicification. In some parts of the ore-body the quartz gangue reaches or exceeds 70%, while in other parts there is very little gangue matter.

The ore solutions met different conditions of temperature and pressure as they ascended, and as a result the mineralization at the higher levels was somewhat different from that below. A section of 800 feet in vertical extent is exposed in the Premier workings. The nature of the ore-body above and below this section can only be conjectured, and will not be discussed here.

The ore at the base of the exposed section is a heavy sulphide, consisting chiefly of pyrite, with a good deal of galena, a little sphalerite and chalcopyrite, and commercial values in gold and silver. Grey copper (tetrahedrite) has not been identified here, but it may be present in small quantities. In the middle of the section, the ore deposited at this time consisted mainly of pyrite, with galena, sphalerite, and a little chalcopyrite and tetrahedrite. As compared with the ore at the base,

chalcopyrite and perhaps galena are less plentiful; tetrahedrite and perhaps sphalerite are more in evidence; and gold and silver values are a great deal higher. Occasional shoots of exceptionally rich ore are encountered. The ore deposited at the top of the section consisted of pyrite chiefly, with a good deal of tetrahedrite and sphalerite. Galena and a little chalcopyrite are also present. Other minerals appearing here are electrum, and perhaps some silver sulphides such as argentite and ruby silver. The ore carries high values in gold and silver.

Very little work has been done so far to determine just what minerals are responsible for the values in gold and silver. This much, however, can be said: At the base of the section, pyrite carries practically all of the gold; in the central zone, pyrite carries perhaps more gold than it does below, and some gold is carried by the sphalerite; at the top of the section, pyrite, sphalerite, electrum, and perhaps tetrahedrite, are the gold carriers. As stated above, sphalerite is fairly abundant here and apparently it carries good values in gold. The values in gold decrease rapidly with depth in the upper 600 feet of the section, and this decrease is explained in part by the disappearance in depth of certain of the gold-carrying minerals.

At the base of the section, again, the low values in silver appear to come chiefly from the galena. In the middle portion, tetrahedrite, and perhaps some silver minerals, are responsible for the rapid increase in silver values. In the upper zone, the greater abundance of tetrahedrite and silver minerals bring the values still higher. Thus we have a rapid change in content of silver from high-grade to low-grade with depth, due in large part to the change in mineralization. Below the base of the exposed section rapid fluctuations in the content of gold and silver are not to be expected.

After the period of dyke intrusion and ore deposition, the Coast Range igneous activity in the area became quiescent, so far as outward signs are concerned. Probably for a considerable time during the Cretaceous period the area was rather low-lying. Uplift followed, however, and the agents of weathering and erosion became active. During the period of erosion the rocks were again faulted, fractured, and shattered. This was probably due to a settling down of certain areas. Fractures cutting across the ore bodies are very numerous.

Disintegration and erosion apparently continued until the Pleistocene period. These agencies stripped the covering of argillites from the main part of the area, unroofed the Coast Range batholith, and cut down deeply into the series of volcanic rocks. The ore-bodies became exposed to the surface and were partially eroded away; but although their upper parts were thus removed, the values did not entirely escape. Part of the silver was carried downward in solution and re-precipitated when these solutions met and reacted with ore under the surface. The fractures and faults afforded easy channels for these downward moving solutions. This secondary action resulted in the deposition of native silver, and probably also of some argentite, ruby silver, polybasite, and other silver minerals. It was carried to considerable depth as some secondary silver is found 400 feet below the present surface. Secondary ores are especially prevalent in fractured vein matter. Secondary enrichment resulted in bonanza shoots of silver ore, as well as in a general enrichment in silver. The decrease in silver values in the primary ore with depth is accentuated by this secondary enrichment in the upper zone, so that the change from high-grade to low-grade silver ore seems rather abrupt.

Pleistocene glaciers came and went, eroding to a variable depth, but probably not to a very great depth in the Salmon River district. The melting of the ice left the district practically as it is to-day — a rugged area covered locally by a mantle of glacial drift. Glacial erosion did not cut away all the secondary ore.

As previously stated, the mine is on the western slope of Bear River ridge, a north-south ridge separating the drainage basins of the Bear and Salmon rivers. East of the mine the ridge rises to an elevation of 6,000 feet above the sea. Cascade creek flows southward one mile west of the mine and joins the Salmon river in Alaska. The bed of the creek west of the mine is 700 feet above sea level.

The structure of the west slope of Bear River ridge is not easy to decipher, since glacial drift covers much of the area, especially in the vicinity of the mine. Coarse volcanic fragmental rocks outcrop on the upper 3,000 feet of the ridge. Below this, down to Cascade creek, the rocks are the green schists of the upper part of the volcanic series, with intruded

sills of porphyry. This side of the ridge appears to be part of the western limb of a northerly striking and northerly plunging anticline. The dip of the limb is steeper than the slope of the hillside. Faults and minor folds complicate the main structure.

## HISTORY OF MINING AND DEVELOPMENT.

The following particulars of the development of the mine have been taken from articles by Mr. Wm. Bunting, from the Minister of Mines' Reports, and from articles by Mr. H. A. Guess.

The chief Premier claims were staked and recorded in the summer of 1910 by Wm. Dilworth and Wm. Bunting. The two main claims, Cascade Falls Nos. 4 and 8, and some adjoining claims, notably the Simpson claim, staked by others, passed into the control of O. B. Bush, who organized the Salmon-Bear River Mining Company. This company began operations in the spring of 1911 and continued work during the summers of 1911 and 1912.

R. G. McConnell, of the Geological Survey of Canada, spent part of the season of 1911 in the Salmon River area. Among other properties mentioned in his report are the holdings of the Salmon-Bear River Mining Co. It is interesting to see how the property was described by a geologist the year after it was staked and before much work had been done. His descriptions are accurate and his advice sound. He says: "Cascade Falls No 4 contains a very wide showing. The schists are silicified, seamed with irregular quartz seams, and impregnated with sulphides for a width of at least 75 feet. Portions of the zone are heavy sulphide and other parts are only slightly mineralized..... The permanence depends on the persistence of the gold and silver values. On the Simpson claim there is another zone of silicified schist carrying locally native silver, high values in silver and some gold from picked samples." McConnell recommended more surface work in order to develop the property.

After lying idle during the summer of 1913, the property was examined by W. J. Rolfe in 1914 in the interests of the Canadian Mining and Exploration Co. The outbreak of the war may have had much to do with the cessation of operations by this company.

Immediately thereafter the property was bonded by Wm. B. Thomson. Work began during the winter of 1914-15 and continued for a year and a half, when operations were discontinued.

In the spring of 1916, Mr. Daly leased the property and mined some ore but made no shipment.

R. K. Neill then examined the property and obtained a bond in March, 1917, in the interests of Trites, Wood and Wilson. Soon after he began work, and in a very short time encountered high-grade gold-silver ore; but very little ore was shipped because of transportation difficulties.

In the summer of 1919, H. A. Guess, of the American Smelting and Refining Co, in conjunction with H. B. Price, engineer for Keith and Untermyer, examined the mine. They purchased a controlling interest for a group of four parties, the American Smelting and Refining Co., Guggenheim Bros., M. C. Keith, and Isaac Untermyer. The American Smelting and Refining Co. have the management of the mine.

Since 1919 the mine has been equipped with water power, auxiliary power by semi-diesel engines for winter months, a 100-ton mill, and a 11½-mile aerial tramway. There are now over two miles of tunnels and drifts, and more than three miles of diamond drilling has been carried out.

The following figures of production are taken from the Minister of Mines' Reports:—

| Year | Ore mined, tons | Gold, ounces | Silver, ounces |
|---|---|---|---|
| 1918 | 26 | 182 | 182 |
| 1919 | 488 | 3209 | 108,285 |
| 1920 | 799 | 2283 | 77,180 |
| 1921 | 18,750 | 40,104 | 1,177,987 |

The daily production at present is 300 tons, of which 100 tons are shipped to Tacoma and 100 tons to Anyox; the remaining 100 tons being milled.

In conclusion, there is one point which the writer would like to bring out, because it is exemplified very well at the Premier. In areas where drift is rather plentiful, heavy sulphide bodies do not as a rule outcrop. They disintegrate and wear down under the weather more rapidly than does a moderately hard rock. This has been recognized in a general way for many years, but in numerous places where prospecting is going on, it seems to have been overlooked. At the Premier mine the outcrops that stand up along the vein are the silicified parts of the ore zone. The heavy sulphide is covered by drift. Stopes above the upper tunnel have broken through to the surface by the side of open cuts as well as between them. Only two feet of drift covered the best ore in these instances. The value of surface work in prospecting, or in developing a prospect, is generally underestimated.

# ROCK DRILL, BIT, AND STEEL SHARPENING AT THE MURRAY MINE.

By E. Hibbert.

Cobalt Branch, 1922.

## Development of the Bit.

The first work undertaken on reopening the Murray mine was the sinking of an inclined shaft (36 deg. from the horizontal), mainly through a hard greenstone. At that time (1913), piston drills were used and the usual western practice of dropping ⅛ in. between gauges was followed, the steel being sharpened by hand.

Soon afterwards a Leyner drill sharpener was installed, but as this made drill bits with a 14 deg. taper, trouble arose immediately—the wear of the gauge being greater than ⅛ in. between steel changes— and for a time it was necessary to have recourse to 3/16 in. changes in gauge. Dollies and dies were made with a 5 deg. taper, but with this the dolly had a tendency to jam in the die and so the taper was altered to 10 deg. The next move was to revert to the 14 deg. taper for dies and dollies, make the bit with this taper, and then flatten the points of the wings to a 5 deg. taper between gauging blocks.

About this time the Carr bit came to our notice, and the idea of utilizing the wing edges of the bit for reaming purposes, and so permitting the use of smaller gauge changes, seemed so important that several single bits were made up on this principle. These, at first, were not correctly formed and so were not very successful, but enough work was done with them to indicate their possibilities.

The writer then made a trip to several of the larger mines in the southwestern States, and to the Michigan copper country.

In the soft porphyry mines of the southwestern States, steel sharpening methods and the form of drill bits are not regarded as important; but in the Michigan copper region a great deal of study has been devoted to these questions and the writer

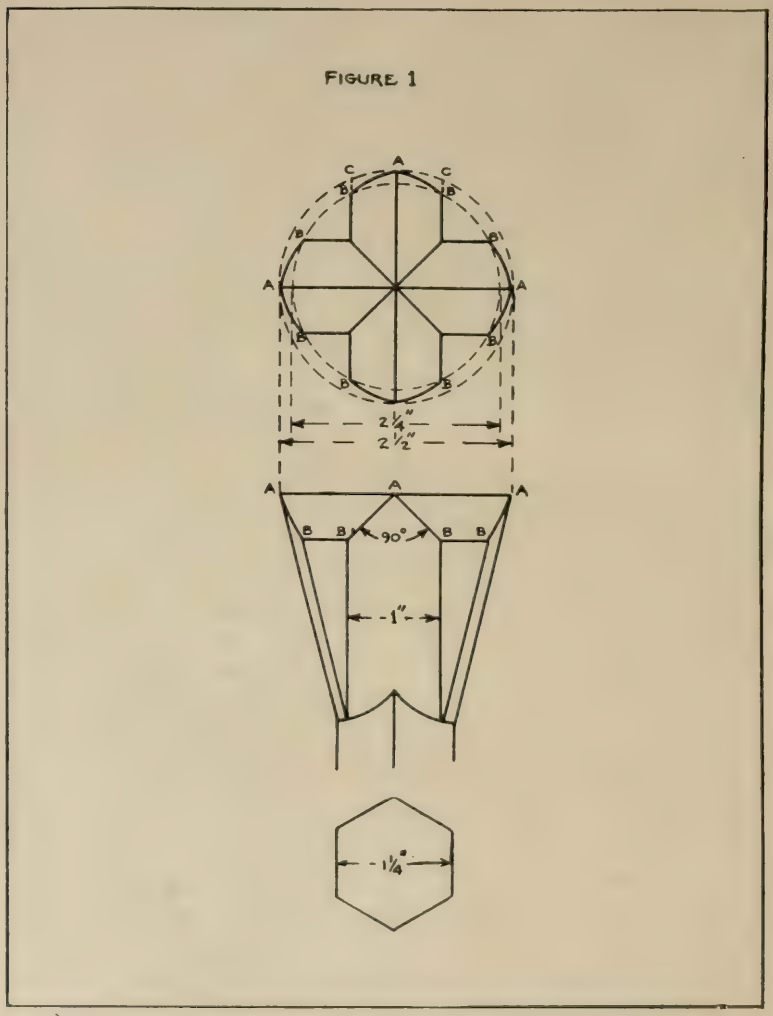



Figure 1

is particularly indebted to Mr. Morrison, master mechanic of the Calumet and Hecla, for much valuable information on the

subject. This trip confirmed our belief in the possibilities of the Carr type of bit, so the problem was again attacked.

The difference between the Carr bit and the type previously made lies entirely in the development of reaming edges. The old type of bit, with 14 deg. dollies and dies, is shown in Fig. 1. It is 2½ in. gauge, with wing one inch thick, and has a 90 deg. cutting edge. It will be noted that, with a new bit, the only portions employed in maintaining the diameter of the hole are the points *A*,*A*,*A*,*A*,—the extreme ends of the cutting edges—and that not until the bit has lost ¼ in. gauge will the points *B*,*B*,. . . . become effective.

The idea of the reaming bit is to make the entire edge, *A* to *B*, take part, from the commencement of drilling, in maintaining the diameter of the hole. To do this the point *A* must be flattened down, or the points *B*,*B*,. . . pulled out, until they occupy the positions *C*,*C*,. . . .

From previous experience and observation it was decided that a 90 deg. cutting edge and a 7 deg. wing taper offered the best possibilities for the hard ground at the Murray mine, and it was decided to first make a bit using 14 deg. dies and dollies and then flatten this on the end to 7 deg. between gauging blocks, using the latter at the same time to flatten the conical surfaces of the wings so that the corners, *B*,*B*, (Fig. 1), would be pulled out to *C*,*C*, and the sharp edges, *AB*, *AB*, thus converted into reaming edges.

In order to determine how much flattening was required, a large-scale drawing was made of a 2½ in. starter with 1 in. wing, 7 deg. wing taper, and 90 deg. cutting edge (Fig. 2). In considering this and the following figures it may be helpful to remember that the points *A*,*A*, lie in a plane at right angles to the long axis of the drill, and that the points *B*,*B*,*D*,*D* lie in another plane parallel to the first, the distance between these two planes being equal to one half the width of the wing (where the cutting-edge angle is 90 deg.).

Travelling from the extreme point *A* of the cutting edge down towards the shank on a 7 deg. taper, a point *D* is arrived

at in the plane $B,B$. Since it is required that the points $B,B$ lie on a circle of the same diameter ($2\frac{1}{2}$ in.) as do the points $A,A$, the curvature to be given to the outside of the wing of

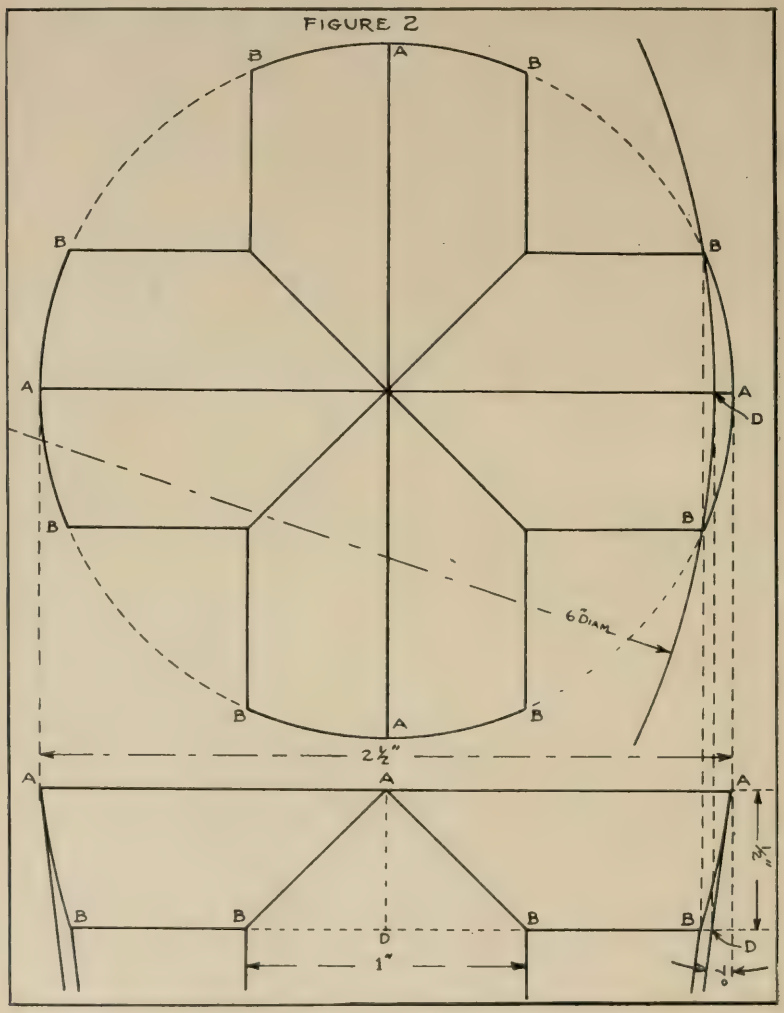



Figure 2

the bit must be such that the points $B,D,B$ all lie on this curve. In this case it was found (by trial on a large-scale drawing) that a 6 in. diameter circle will satisfy the above conditions.

If, therefore, a bit is made with 14 deg. taper dies and dollies, and flattened at the end between gauging blocks having cylindrical surfaces (6 in. diameter curvature) which are inclined at 7 deg. to the long axis of the bit until the gauge of the bit is 2½ in., the points $A,A,B,B$, as seen in plan, will all lie on a 2½ in. circle, and they will all play their part in reaming out and maintaining the diameter of the hole. Fig. 7A shows a

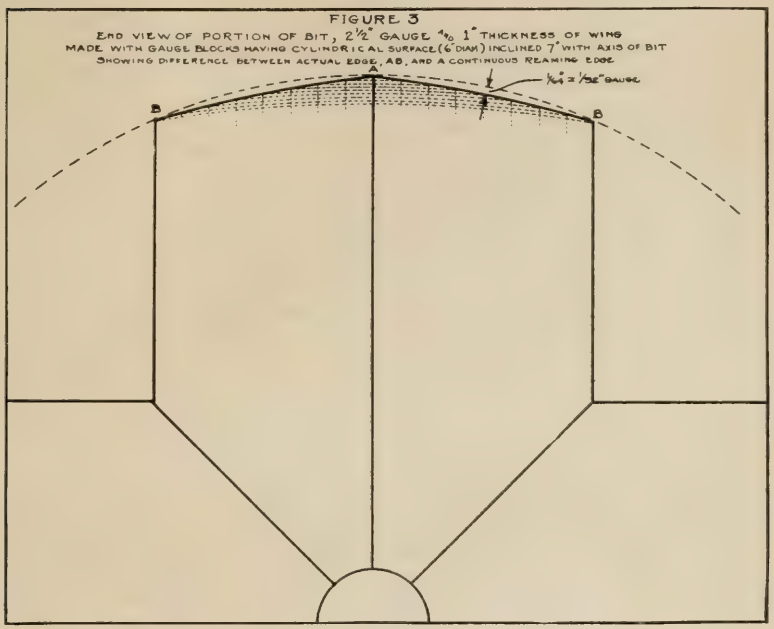


FIGURE 3
END VIEW OF PORTION OF BIT, 2½" GAUGE, ⅞ 1" THICKNESS OF WING
MADE WITH GAUGE BLOCKS HAVING CYLINDRICAL SURFACE (6" DIAM) INCLINED 7° WITH AXIS OF BIT
SHOWING DIFFERENCE BETWEEN ACTUAL EDGE, AB, AND A CONTINUOUS REAMING EDGE
A
B
B

**Figure 3**

bit being flattened between two such gauging blocks. Detail of gauging blocks designed for use in the Leyner sharpener is shown in Plate 1.

For a time it was assumed that, if this condition was satisfied, all parts of the curved line $AB$ would aid in reaming the hole from the commencement of drilling; but in developing this line in plan (Fig. 3) this was found not to be the case. The variation, however, is small, and after the gauge has worn 1/32 in. all portions of the edge help in the reaming, as compared with a necessary wear of ¼ in. gauge before this condition is reached for the bit illustrated in Fig. 1. It will be shown

below (Fig. 5) that in the case of a bit with 1¾ in. diameter and ⅝ in. wing, the variation of the actual bit from a true reaming edge is only 1/128 in.

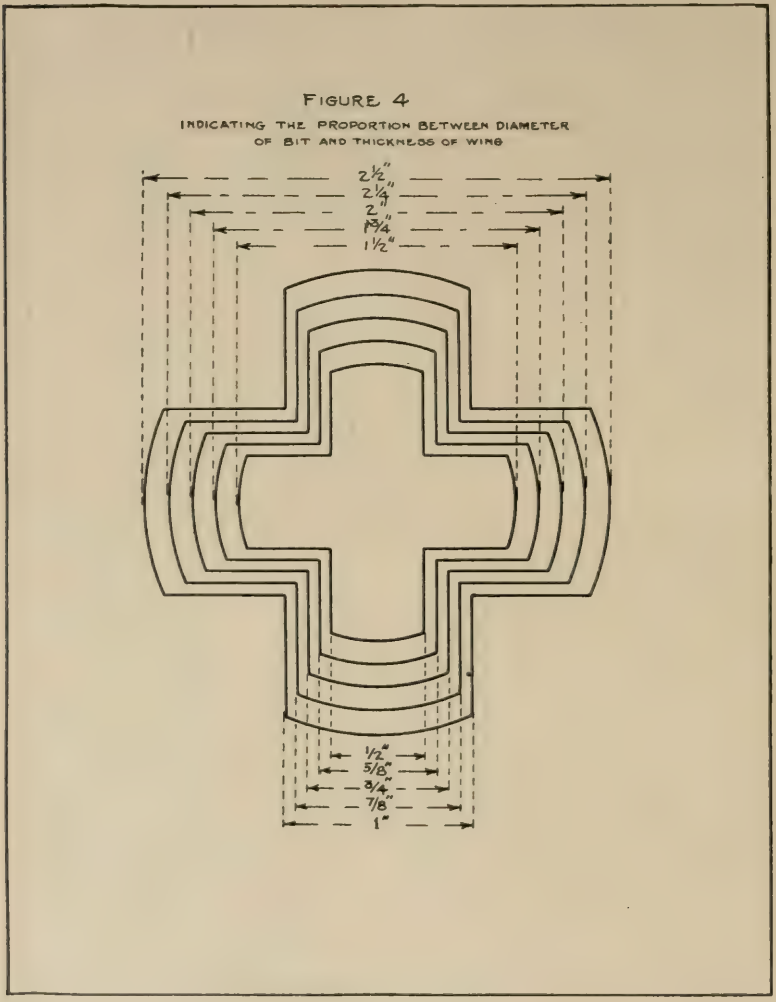



**Figure 4**

The next consideration was the determination of the most effective ratio between thickness of wing and diameter of bit. It was evident that the gauging blocks referred to above would

not give correct results for a 1¾ in. diameter bit with 1 in. width of wing, 90 deg. cutting edge, and 7 deg. wing taper; and as such dimensions seemed out of proportion, it was decided to make the bits according to the following specifications (see Fig. 4):—

| | |
|---|---|
| 2½ in. diameter, | 1 in. wing |
| 2¼ in. " | ⅞ in. " |
| 2 in. " | ¾ in. " |
| 1¾ in. " | ⅝ in. " |
| 1½ in. " | ½ in. " |

With these dimensions, there is an approximately constant ratio between diameter of bit and thickness of wing, and it was found that the one set of gauging blocks, made as shown in Plate 1, could be used for finishing or gauging any bit within the above range (see Fig. 5). When a bit is to be sharpened, a stop plate, or "gauge plate", of suitable size, is slipped over

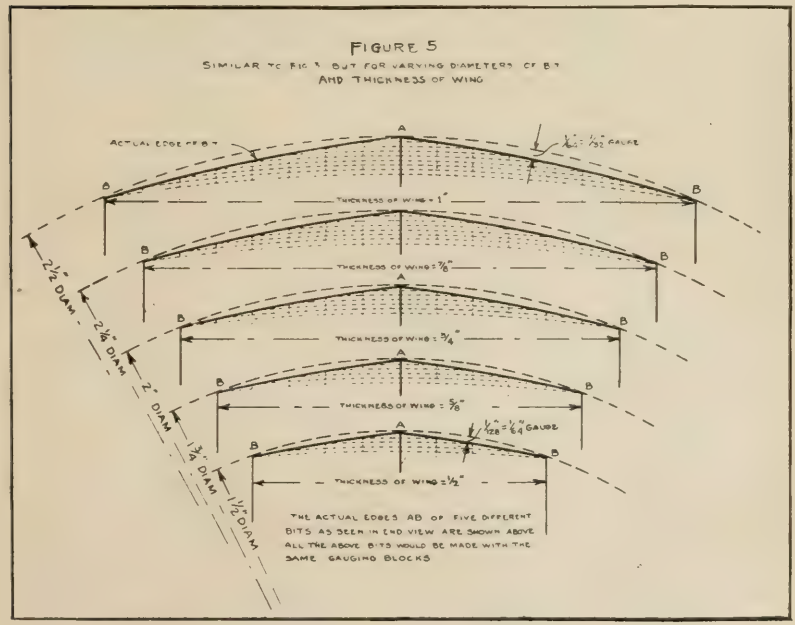


FIGURE 5
SIMILAR TO FIG 1 BUT FOR VARYING DIAMETERS OF BIT AND THICKNESS OF WING
ACTUAL EDGE OF BIT
A
B
THICKNESS OF WING = 1"
THICKNESS OF WING = ⅞"
THICKNESS OF WING = ¾"
THICKNESS OF WING = ⅝"
THICKNESS OF WING = ½"
2½" DIAM
2¼" DIAM
2" DIAM
1¾" DIAM
1½" DIAM
THE ACTUAL EDGES AB OF FIVE DIFFERENT BITS AS SEEN IN END VIEW ARE SHOWN ABOVE ALL THE ABOVE BITS WOULD BE MADE WITH THE SAME GAUGING BLOCKS

**Figure 5**

the bottom gauging block to hold the bit at the proper point along the tapered cylindrical surface (See Fig. 10 and Plate 1).

Although a 7 deg. wing taper was considered advisable at the Murray mine, the most suitable slope will be determined, at any place, by the type of ground met with. If an 11 deg.

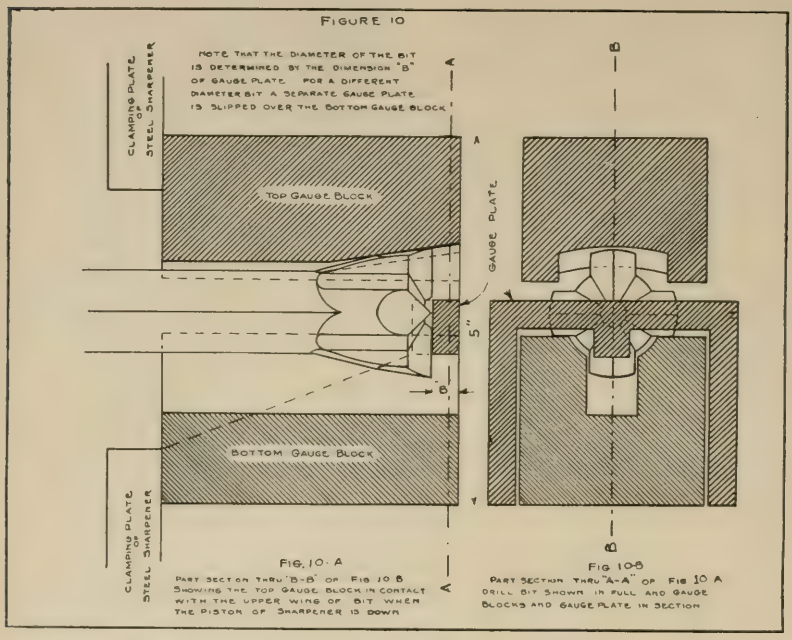


FIGURE 10
NOTE THAT THE DIAMETER OF THE BIT IS DETERMINED BY THE DIMENSION "B" OF GAUGE PLATE. FOR A DIFFERENT DIAMETER BIT A SEPARATE GAUGE PLATE IS SLIPPED OVER THE BOTTOM GAUGE BLOCK
CLAMPING PLATE OF STEEL SHARPENER
TOP GAUGE BLOCK
GAUGE PLATE
5"
B
BOTTOM GAUGE BLOCK
CLAMPING PLATE OF STEEL SHARPENER
FIG. 10-A
PART SECTION THRU "B-B" OF FIG 10 B SHOWING THE TOP GAUGE BLOCK IN CONTACT WITH THE UPPER WING OF BIT WHEN THE PISTON OF SHARPENER IS DOWN
FIG 10-B
PART SECTION THRU "A-A" OF FIG 10 A DRILL BIT SHOWN IN FULL AND GAUGE BLOCKS AND GAUGE PLATE IN SECTION

**Figure 10**

taper is employed, then for a 2 in. bit with ¾ in. width of wing, a gauging block with a perfectly flat surface would be called for (see Fig. 6A); whereas for a similar bit, but with a 14 deg. wing taper, the gauge blocks should be convex if it is desired to develop reaming points in addition to the extreme ends of the cutting edges. (See Fig. 6B).

When the investigation had reached this point, a milling cutter, with cutting face ground to 6 in. diameter, was rigged up on a lathe and the making of gauging blocks commenced (see Plate 1). At first the upper and lower blocks were made the same shape, and fairly good bits resulted. On measurement with the calipers, however, they were found to exhibit

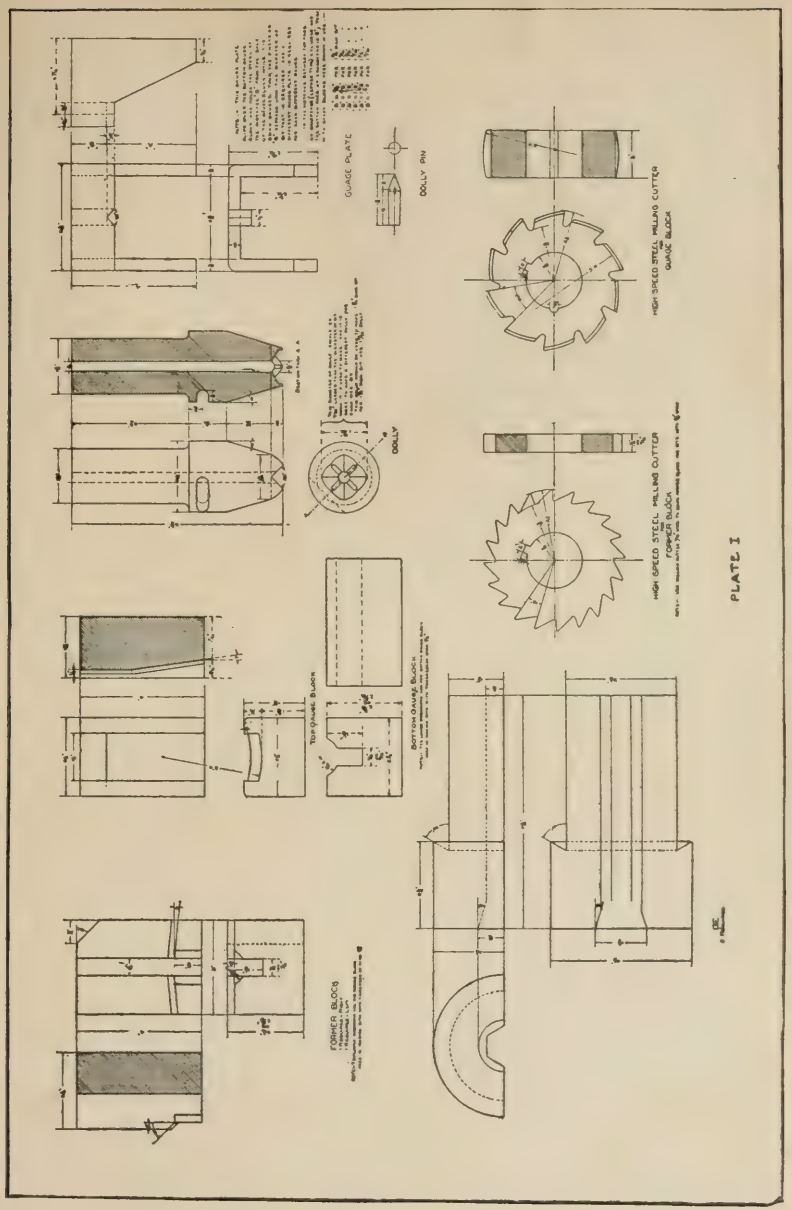

GUAGE PLATE
DOLLY PIN
HIGH SPEED STEEL MILLING CUTTER
GUAGE BLOCK
DOLLY
HIGH SPEED STEEL MILLING CUTTER
FORMER BLOCK
PLATE I
FORMER BLOCK

**Plate 1**

differences, and these were traced to the fact that the wings of the bit were not being held squarely with the gauging blocks (see Fig. 7B). In gauging a 2-in. bit, for example, it was found that, if the blacksmith rotated the steel so that the cutting edges were inclined 5 deg. from the vertical and horizontal position, the distance $BB$ was 3/32 in. greater than $B'B'$, although only 3/64 in. greater than the correct gauge. However, as at this time efforts were being made to use a 1/32 in.

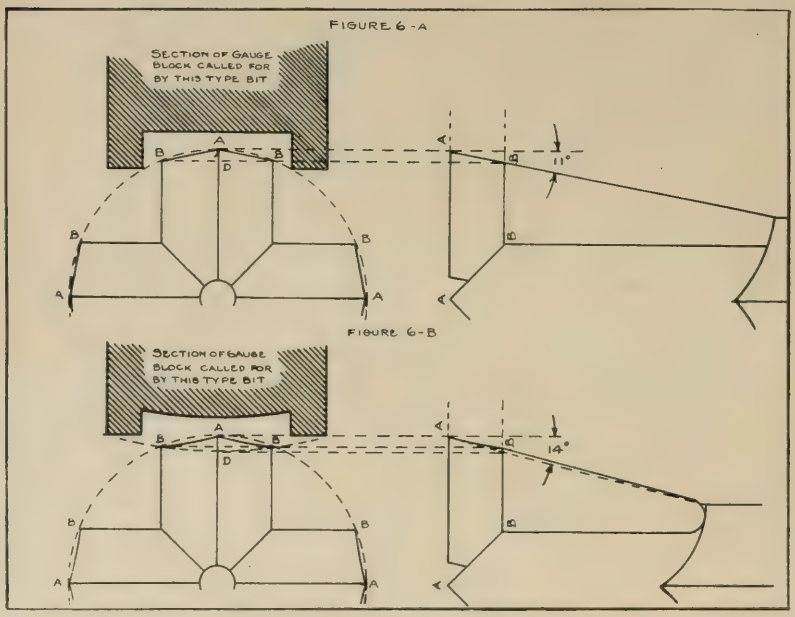



**Figure 6A, 6B**

drop in gauge between steel of about one foot difference in length, this condition had to be corrected. This was done, as shown in Fig. 7C, by using a former block for the lower gauging block, in order to hold the wings of the bit square with the gauging blocks and prevent the blacksmith from rotating the drill when gauging the bit.

With the bit as then made, there were sharp re-entrant angles at $EF$ (Fig. 8A), from which occasionally, though rarely, cracks started in the steel. These sharp angles were eliminated by turning off the corresponding edges of the dolly, and a bit as shown in Fig. 8B was made.

The size of the drill steel used at the Murray mine is 1⅛ in. hollow hexagonal. When sharp angled former blocks are used in making the bit with this steel, as shown in Fig. 8B (⅝ in. wing), the material at *EF* is driven towards the centre and so tends to close up the hole. To overcome this trouble, the former blocks were hollowed out to 1 3/16 in. diameter, and the re-entrant angle *BFB* was partly done away with. The bit, as now made, is shown in Fig. 8C; see also

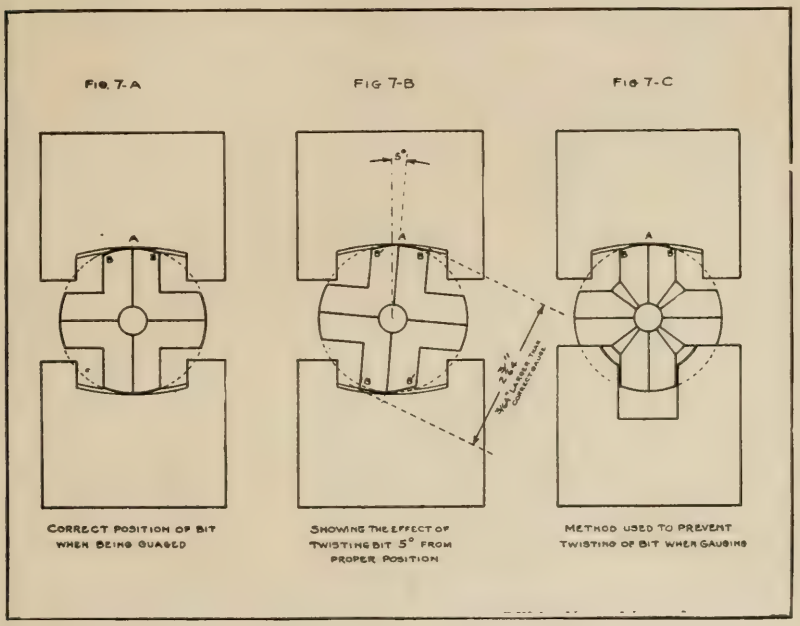


FIG. 7-A
FIG 7-B
FIG 7-C
CORRECT POSITION OF BIT WHEN BEING GUAGED
SHOWING THE EFFECT OF TWISTING BIT 5° FROM PROPER POSITION
METHOD USED TO PREVENT TWISTING OF BIT WHEN GAUGING

**Figure 7A, 7B, 7C**

Fig. 11. With this bit it is considered that, when drilling in the ore body, a 14-foot hole can be put down using a 1¾ in. starter with the rest of the steel made to the gauges shown in Fig. 11.

Owing to the Murray mine being closed down, this newly developed bit has not yet received a proper test; but a winze was sunk, partly in the ore and partly in the greenstone footwall, using this size steel. When in the ore no difficulty was experienced in changing steel, but in the greenstone there was

trouble, particularly with the longer steels. For a time, whilst in the ore 1⅝ in. starters were used when drilling an 8-foot hole, but in the greenstone 1¾ in. starters proved necessary.

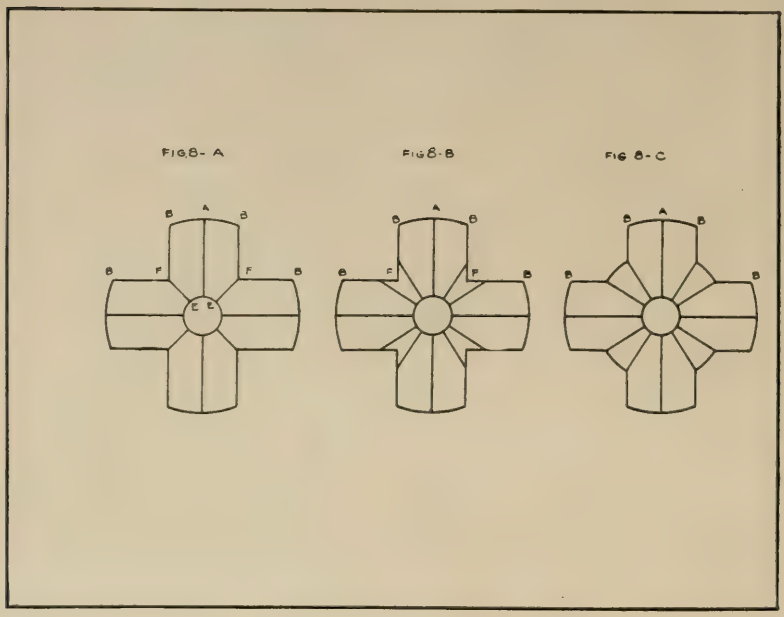


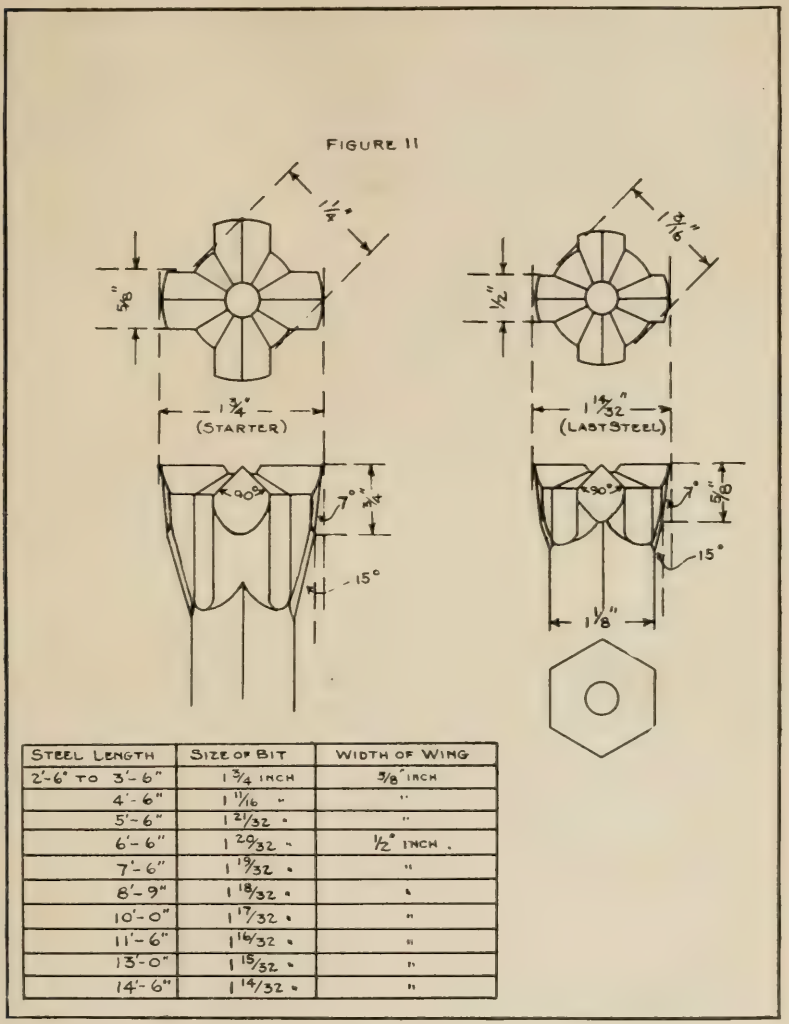

**Figure 8A, 8B, 8C**

## DRILL SHARPENERS AND TOOLS.

In order to sharpen correctly steel with small differences in gauge, it is essential that tools used in the drill sharpener be kept in first-class order. Also, in selecting a drill sharpener it is advisable to consider whether the various dies, dollies, former blocks and gauging blocks, can be easily and correctly re-adjusted after being re-surfaced, or whether it will be necessary to replace these parts when wear takes place.

The first drill sharpening machine installed at the Murray mine was a Leyner. At that time, solid steel was being used, with a 2¾ in. starter. The Leyner machine, as then developed, was a little slow for sharpening this steel, so when it became necessary to instal another sharpener a Sullivan was selected. This machine reduced the time of making the big bits, but no

method was developed for readily taking care of the wear on gauging and former blocks, and for this reason the machine fell into disuse. At the present time all the drill sharpening




| STEEL LENGTH | SIZE OF BIT | WIDTH OF WING |
|---|---|---|
| 2'-6" TO 3'-6" | 1 3/4 INCH | 5/8 INCH |
| 4'-6" | 1 11/16 " | " |
| 5'-6" | 1 21/32 " | " |
| 6'-6" | 1 20/32 " | 1/2" INCH . |
| 7'-6" | 1 19/32 " | " |
| 8'-9" | 1 18/32 " | " |
| 10'-0" | 1 17/32 " | " |
| 11'-6" | 1 16/32 " | " |
| 13'-0" | 1 15/32 " | " |
| 14'-6" | 1 14/32 " | " |

Figure 11

machines in use are Leyners, four being set up and one kept as a spare in case of breakdowns. If some method could be devised whereby the steel sharpener could operate the Leyner

machine by means of a foot pedal, as with the Sullivan, it is probable that the speed of drill sharpening with the first-named machine would be increased at least 10%.

Plate 1 shows the various tools which have been developed at the Murray mine for use in the Leyner sharpener in making the present bit. With regard to wear, it has been found that, before requiring to be re-surfaced, former blocks and gauging blocks can be used in making about 6,000 steel, and dollies about 12,000; dies seldom need replacing.

It will be seen that the proper care of drill sharpener tools necessitates a large amount of machine shop work. A milling machine equipped with proper cutters would do most of this work quicker and more economically than any other machine and should prove a good investment where many drill sharpeners are in use.

## PUNCHES.

For punching the hole in the shorter steel, a Waugh punch is attached to the drill sharpener. This works satisfactorily on short steel but is not convenient to use with the longer ones, slowing up operations. For the longer steel, a punch made from an old Holman plugger machine is used. This is placed between the furnace and the drill sharpener, and is so arranged that pressure of the drill steel on the point of the punch starts the punching operation.

## FURNACES FOR HEATING THE DRILL STEEL.

Each sharpener has its own oil-fired furnace, and one furnace in addition is used solely for tempering purposes. The furnaces, which work very successfully, were made from drawings of furnaces used at the Hancock mine, Hancock, Michigan, and these in turn, I believe, were modelled on those in use at the Calumet and Hecla. The furnaces are shown in Fig. 12. The double front, through which a stream of cold air was supposed to blow, is no longer used. Failure of the cooling water supply has to be closely watched, as the cooling jacket is easily burnt out.

Two spare furnace bottoms are kept on hand, and when the brick lining of a furnace burns out it is exchanged for a spare bottom and then relined at leisure.

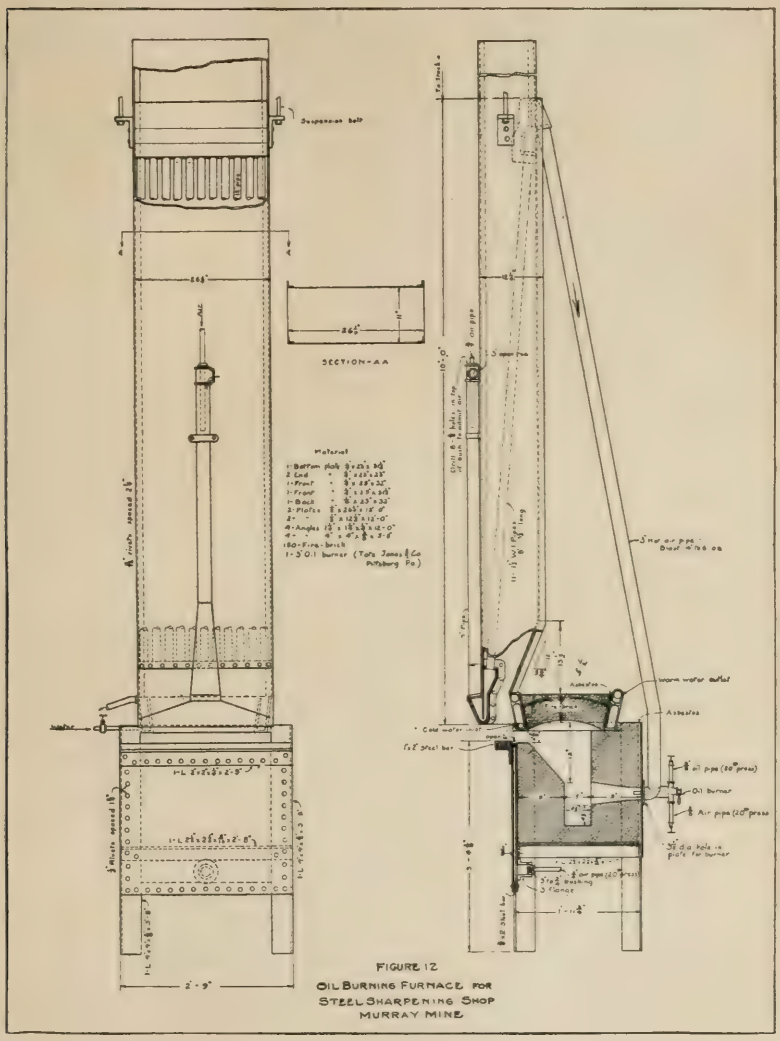



Figure 12

After about forty shifts, the ordinary fire brick lining of a furnace fails through being slagged, probably from iron oxide

scales from the steel. With a view to extending the life of the lining, chrome brick is being tried on two furnaces, but these have not been in use a sufficient time to prove whether the increase in life is sufficient to justify the higher cost. Slagging has been largely overcome, but the surface of the brick is inclined to shell off.

No automatic pyrometric arrangements are used to control the furnace heat, and it is a constant struggle to prevent

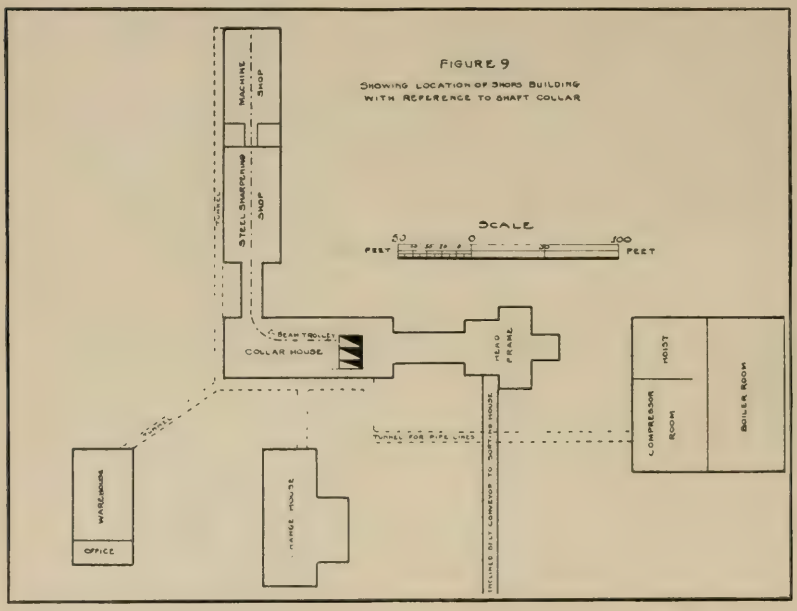



**Figure 9**

the operators running the furnaces too hot. It has not yet been found possible to fix on the right heat and then lock the oil supply valve, because the fuel oil supply is placed some 100 feet away from the blacksmith shop, to which it is fed by compressed air; and variations of temperature in the outside atmosphere, particularly in winter, considerably affect the viscosity of the oil. Attempts have been made to overcome this by heating the oil tank and the passages to the shop by steam pipes. Another source of trouble is due to the fact

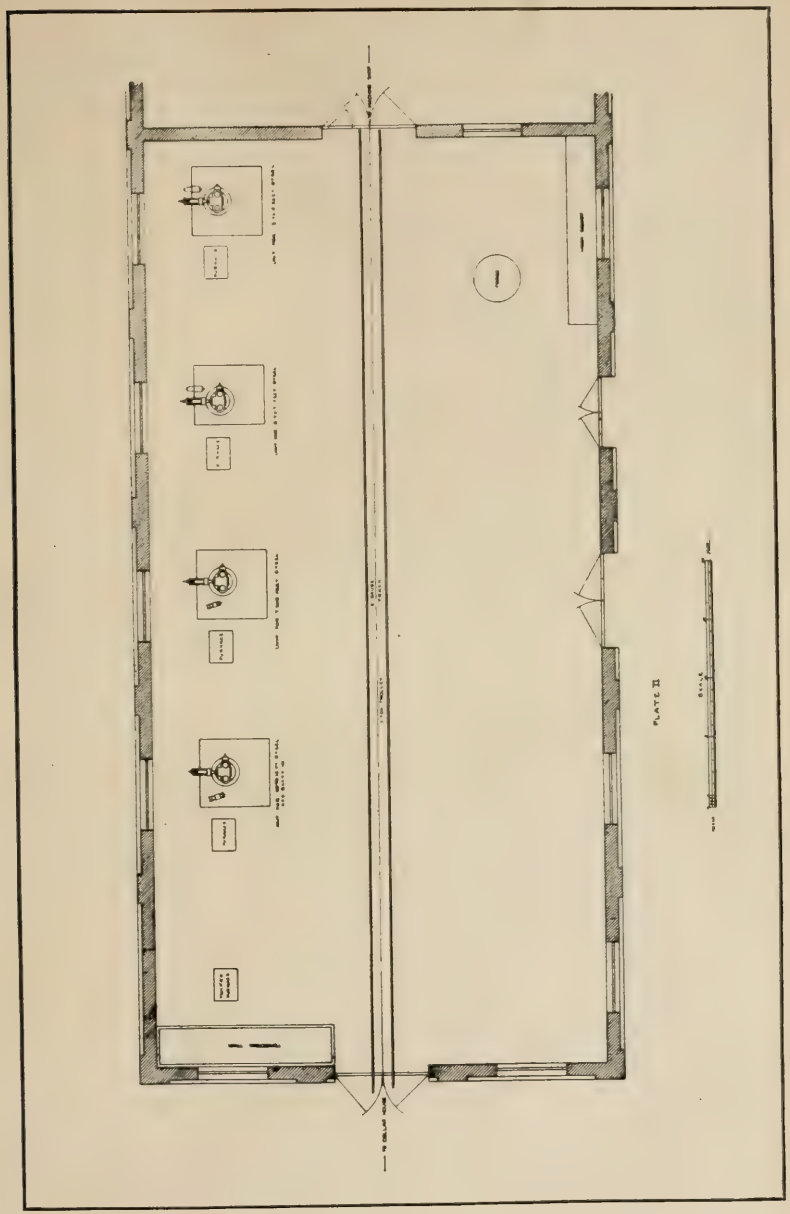


Plate 2

that the fuel oil, as received in car lots from time to time, is apt to vary considerably in character

The consumption of fuel oil per eight-hour furnace shift averaged 16 gallons, and 18½ steel were sharpened, tempered, and shanked per gallon of oil.

## ARRANGEMENT OF BLACKSMITH SHOP.

Fig. 9 shows the position of the steel sharpening shop relative to the collar house, and Plate 2 shows the arrangement of the various machines in the shop.

The operation of the shop is as follows: The dull steel from the mine is classified according to length, steel up to 4 feet going to sharpener No. 1; from 5 feet to 7 feet to sharpener No. 2; from 7 feet to 9 feet to sharpener No. 3; and everything above 9 feet to sharpener No. 4. Each sharpener is then set to sharpen a certain size steel, and all steel of this size are sharpened at one time; dollies, gauge plates, and former blocks, are then changed and all the steel of another length sharpened. In order to allow for the flattening of the points of the bit in the gauging blocks, dollies 3/32 in. larger in diameter than the finished bit are used.

After being sharpened, the steel are placed on trucks or on slings made of plate iron. The slings are picked up by travelling chain blocks and carried to the tempering furnace. After tempering, they are placed on trucks and run out to the collar house. The number of steel sharpened per drill sharpener shift is 374.

## SHANKING STEEL.

Tappet machines are used throughout the mine, so no lugs are necessary on the steel. The ends of the steel are brought to gauge and rounded off, using suitable dies and dollies, in sharpener No. 4. The shanks are then heated cherry red and plunged, the ends cleaned, and the temper afterwards drawn by placing about fifteen of them at a time between two red-hot plates of iron (1 by 4 by 20 in.). When the colour on the ends of the shanks reaches what the blacksmith calls "sky-blue", the steel is pushed back and allowed to cool.

No oil tempering is used, the above method, although a little slow, having proved satisfactory.

In conclusion the writer wishes to thank Mr. H. L. Roscoe, Superintendent of the Murray mine, for his assistance in the design of the blacksmith shop and throughout the various stages of experimentation leading to the production of the present drill bit; and also Mr. W. T. Shellard, who had charge of the drill sharpening shop at the Murray mine, and to whom many of the improvements in the drill bit and drill sharpener tools are due.

# DROP SHAFT SINKING.

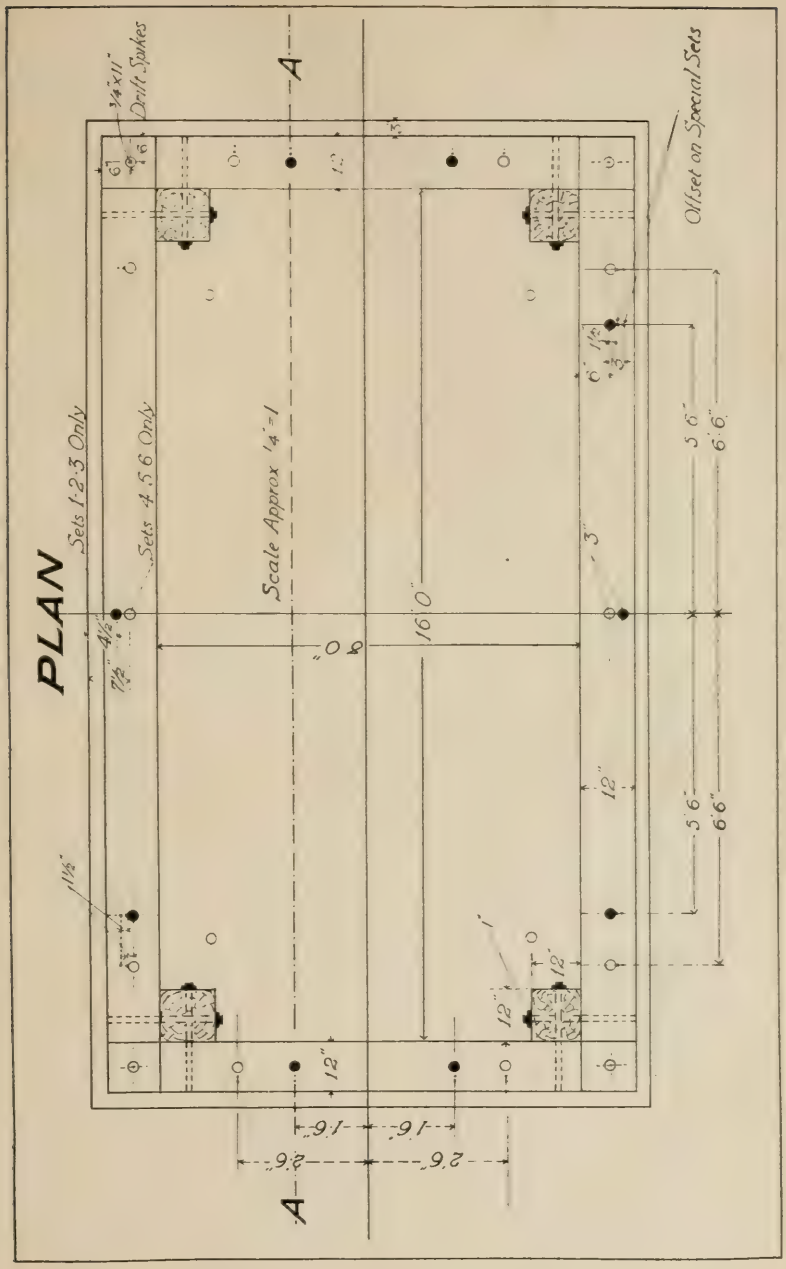
By A. A. MacKay.

Annual General Meeting, Ottawa, March, 1922.

A solid timber drop shaft was successfully sunk on the Cuyuna Iron Range, at Woodrow, Minnesota. Several 'lath shafts' had previously been attempted on the Cuyuna Range under similar conditions, but were usually unsuccessful and resulted in financial loss. One concrete drop shaft had been put down by the compressed air method. The operation was slow and expensive. After considering the results of shaft sinking operations in this district it was thought advisable to sink a solid timber drop shaft.

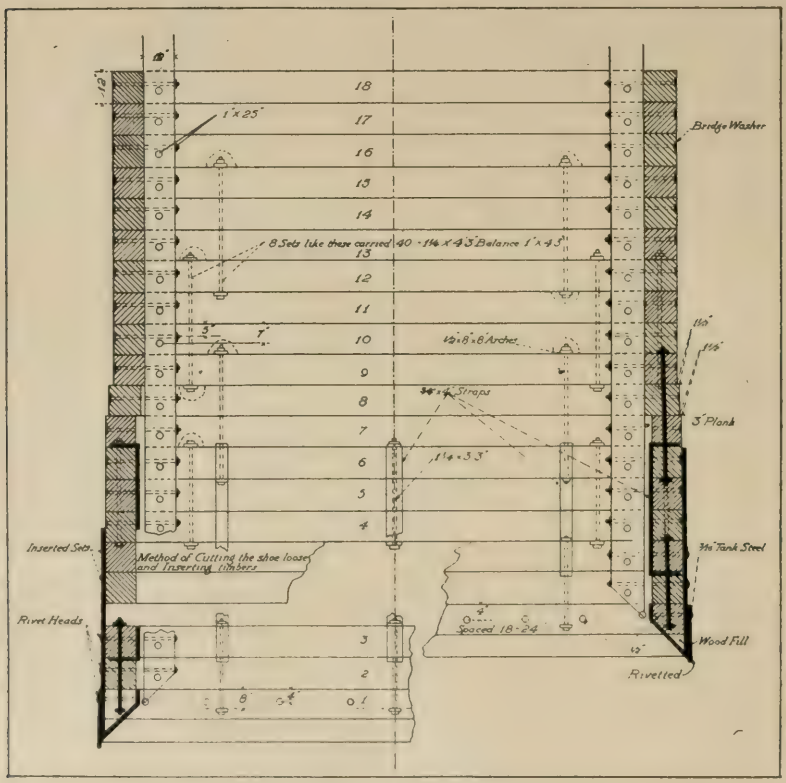
The completed vertical shaft was to be 6 by 16 ft., inside measurements, and was to have a normal capacity of one thousand tons per day. However, to allow for alignment, the drop shaft was made 8 by 16 ft. inside measurements.

The skin friction on the outside of a drop shaft often becomes so great that it is impossible to get enough weight on the unexposed portions of the shaft to overcome it. To be prepared for this emergency, the three bottom sets and the shoe were built independently and connected to the balance of the shaft by ¾ by 4 in. strops so constructed that they could be easily cut, allowing the shoe to be forced ahead of the main shaft by means of jacks and wedges. A 6 ft. by 3/16 in. tank steel apron was bolted to sets Nos. 1, 2 and 3, and extended three feet above set No. 3. This apron was for the purpose of keeping the sand back while the shoe was being forced downward and to give the necessary space to insert sets under set No. 4, as shown in Plate I. In this particular case it was not necessary to force the shoe ahead or to cut the strops, as the shaft dropped two feet below the sand into the altered rock. The shaft was constructed of


PLAN
Sets 1-2-3 Only
Sets 4-5-6 Only
Scale Approx 1/4"=1
Offset on Special Sets
Drift Spikes
A
A
16'0"
9'0"
12"
5'6"
6'6"
9'1"
9'1"
2'6"
2'6"

12 by 12 in. Oregon-fir, surfaced. It was made water-tight by placing two ribbons of oakum between each set. The cost of the timber, in lengths to forty feet, was approximately $21.00 per M., f. o. b. mine. The cost of 3 by 12 in. planking was approximately $19.00 per M., f. o. b. mine.

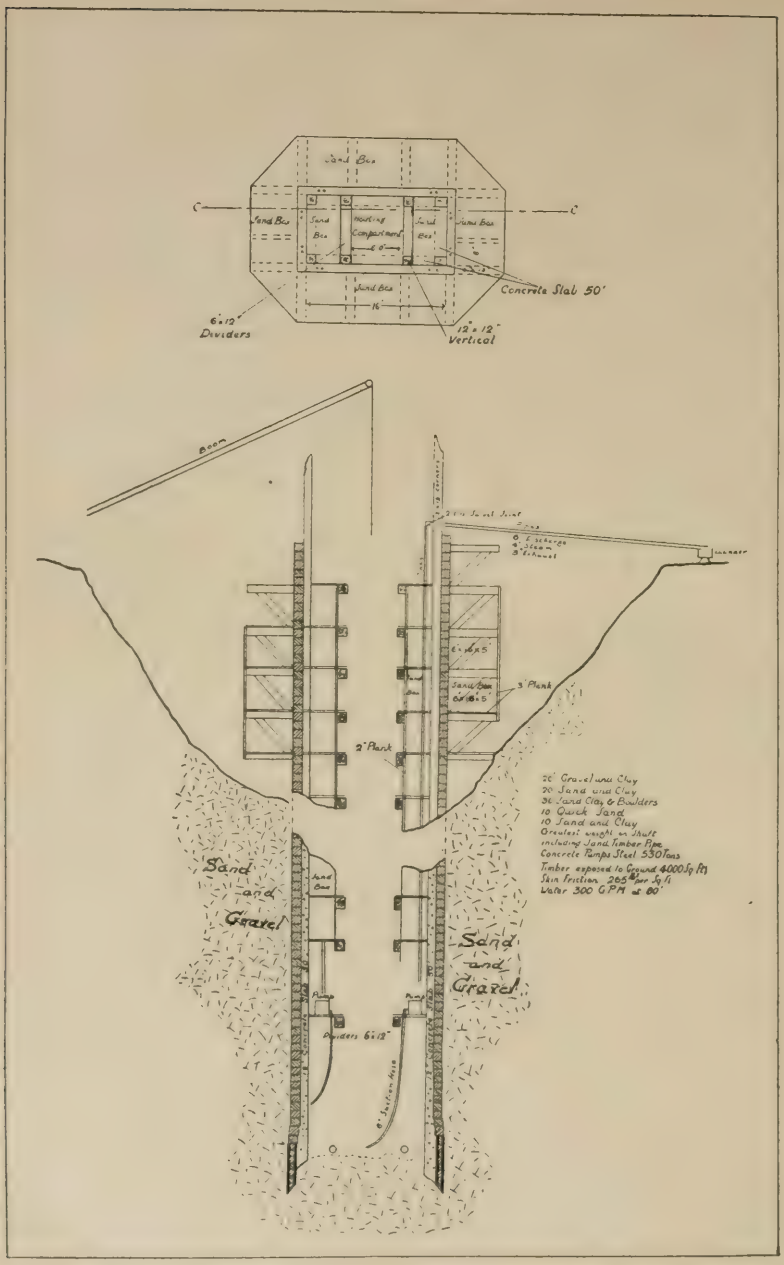
The construction of the shaft is shown in detail in the accompanying Plates and can be easily followed. The lower seven sets were six inches wider in dimension than the main part of the drop shaft. Set No. 8 was three inches wider.




This gave the lower part of the shaft a bell shape. In clay or hard ground this six-inch space relieved the skin friction, while in cases of sand boils, or runs from the sides, the downward pressure on the bell shaped section aided in carrying the shaft downward. The framing was done on the surface

near the shaft site. The shoe and the bell part of the shaft, consisting of eight sets, were constructed and bolted together on the surface. The members of each set were tied together with 3 by 11 in. drift spikes. The sets were tied together with 1¼ in. bolts, except sets Nos. 3 and 4, which were tied together with ¾ by 4 in. strops as shown in Plate I. The second set of bolts, from the bottom were put in after the timbers were in place. This was necessary as the bolts had to go through the strops and the timber. A hand hole was made in set No. 3 under the bolt so that the nut could be held while being tightened. All other vertical bolts were 1¼ in. by 4 ft. 3 in., and were set with head down. The timbers were worked down on the bolts and drawn together with the nuts. The bolts were started every third set, i.e., six, nine, twelve, etc. By this arrangement each set was bolted to the one above and to the one below. Steel plate washers ½ by 8 by 8 in. were used on all vertical bolts. Cast iron bridge washers were used on all other bolts. A derrick with an extra long boom, set well back from the shaft site, was used for hoisting. A headframe was not possible owing to the caving ground near the shaft. When the shaft was completed the ground within a radius of thirty feet had caved to a depth of thirty feet at the shaft. A hole twelve feet deep was excavated at the shaft site with horse scrapers. The seven completed sets were lowered into the excavation with jacks and skids, five sets more were placed, and the forty-foot vertical corner members placed with the derrick and bolted to the twelve sets. Concrete was poured into the spaces between the corner members, which gave extra stiffness to the shaft.

The miners started to dig in the shaft on April 6th. At a depth of 16 feet water was encountered and digging was stopped while enough sets were added to place the pumps in safety. The pumps, two No. 10 Camerons, bored and bushed, were placed twenty feet above the lower set. Owing to the sand boils it was necessary to have the pumps as high as practicable in the shaft. The pumps had been bushed with phosphor-bronze bushings. This was necessary on account of the extreme wear caused by the sand. Extra bushings were kept on hand and could be installed rapidly. When sinking


Sand Box
Sand Box
Sand Box
Working Compartment
Sand Box
Sand Box
Sand Box
16'
Concrete Slab 50'
6"x12" Dividers
12"x12" Vertical
Boom
3" Plank
2" Plank
Sand and Gravel
Sand and Gravel
Pumps
Boulders 6"x12"
20' Gravel and Clay
20' Sand and Clay
30' Sand Clay & Boulders
10' Quick Sand
10' Sand and Clay
Greatest weight in Shaft including Sand Timber Pipe
Concrete Pumps Steel 530 Tons
Timber exposed to Ground 4000 Sq Ft
Skin Friction 265 lbs per Sq ft
Water 300 G.P.M. at 60'

through the fine sand the bushings would last about two weeks, the pumps would have to be packed every hour, and new valves were necessary every day. The end of the suction hose and screen could be raised and lowered by means of rope and blocks. In case of a sand boil, one miner would stand by to keep the hose from getting choked or buried in the sand.

The pipe lines were carried as shown in Plate 3. The pipe was cut in 10-ft. lengths, flanged at both ends and added at the top of the shaft as the latter dropped. The variation in height and length was taken care of by a swivel joint made with two L's. The average flow of water was about 300 gallons per minute. One of the pumps could take care of this flow, thus allowing time to add pipes and pack pumps, etc.

When possible, the miners would dig under the cutting edge of the shoe, but the greater part of the time the sand in the shaft was from two to six feet above the cutting edge, and in very fine sand up to fifteen feet above the cutting edge. At such times the sand would be shoveled from the centre when the shaft was dropping vertically, or from the sides when it was desired to start a sand run from one side or the other in order to bring the shaft back into a vertical plane. Small sand boils could generally be checked by piling cement sacks filled with sand on the boiling area, and by driving oak planks past the shoe. However, the only sure way of stopping sand boils was to have weight enough on the shaft to carry it downward during the boil and thus automatically shut it out. The downward pressure against the projecting shoe during a boil aided materially in carrying the shaft down. During heavy boils the shaft would at times drop as much as six inches in a few minutes. The shift bosses and miners watched the plumb bobs which were kept suspended from the top to see that the shaft did not get out of line or out of a vertical plane.

The chief sources of weight on the shaft were timbers, bolts, steel, pumps, pipe, concrete, and sand. The sand boxes were started in the shaft at set No. 30. The interior sand boxes were 5 ft. high by 4 ft. wide by 8 ft. long, and were continuous to the top. This gave 64 cubic feet of sand to the foot in height. When the shaft was completed there was

60 ft. of sand boxes containing 3,840 cubic feet of sand. The outside boxes were started when the shaft was down 50 feet. These boxes were generally carried 20 ft. high and consisted of four independent boxes, one above the other. When the shaft dropped to where the lower boxes touched the grounp these boxes were removed and others added at the top to take their place. The outside boxes contained about 5,360 cubic feet of sand.

When the shoe was passing through clay and boulders much trouble was caused by boulders getting under the cutting edge of the shoe. In such cases the cutting edge was generally in sight and the boulders could be worked through into the shaft. The largest boulder encountered was three feet in diameter. After passing through seventy feet of stratified material, consisting of layers of gravel, clay, sand, boulders and clay, and sand and clay in beds from two to ten feet thick, the shaft went through ten feet of quick-sand, ten feet of fine sand and clay, and two feet of altered slate.

When the shoe was passing through the quick-sand the more compact material above settled against the walls of the shaft causing a heavy 'drag'. The sticky clay increased the skin friction very much. The upper four feet of the underlying rock was very soft and had the appearance of soft clay. The altered rock made a very good seal, shutting out all the water and sand. Before the seal could be strengthened with concrete on the inside and outside of the shoe, the sand and water broke through in a small hole about six inches in diameter and filled the shaft with sand to within thirty feet of the top, covering the pumps and all the tools. This sand was shoveled out in about four days. The hole under the shoe was found sealed with boulders and clay. The shoe was cemented in and no more trouble encountered. The weight was taken off the shaft on July 1st. The shaft dropped on an average of one foot per day. When completed, it had a nine inch bow in the centre. The lower edge of the shoe was vertically under the top of the shaft. The rock shaft was started in the proper line under the shoe. After sinking in the rock for fifteen feet it was found to be hard enough to hold bearers for the shaft above.

The miners and pumpmen worked in eight hour shifts. The balance of the men worked ten and twelve hour shifts.

The force list included 3 shift bosses, 12 miners, 3 pumpmen, 2 bucket tenders, 2 hoist and firemen, 1 mechanic, 1 blacksmith, 1 blacksmith helper, 3 carpenters, 1 carpenter helper, and 1 timekeeper.

The greatest weight on the shaft at any time was 530 tons. The greatest amount of skin friction was approximately 265 pounds per square foot.

## DISCUSSION.

MR. S. N. GRAHAM: Mr. MacKay has given us such a good description of drop shaft sinking that it is difficult for one not experienced in this type of work to make much comment. So far as I know there have been no shafts sunk by this method in Canada, and I think we can consider ourselves fortunate in not having been called upon to use them. Should such a problem ever arise I am sure that the man tackling it will thank Mr. MacKay for having recorded his experience in the TRANSACTIONS of the Institute.

Most drop shafts have been of concrete, and I am tempted to ask Mr. MacKay if his is the first timber drop shaft. In his use of timber he adopted a very ingenious method to make it water-tight and it would be interesting to know whether straining, causing the "bow" he mentions, opened up any joints. Even in such a case it seems to me probable that these would be soon plugged with fine sand and silt unless the ribbons of oakum had been pushed out of place.

Mr. Mackay does not say whether the concrete lining was reinforced or how it was anchored to the timber. While he was able to keep the alignment of his shaft very true, yet it seems that there would be danger of the straining cracking this long thin slot and allowing pieces to drop out unless it were well reinforced.

I would like to refer to the excellent engineering practice in providing for contingencies. The proposed method to continue sinking, should the shaft be hung up by skin friction, seems designed to meet all requirements. What would he do when he reached the limit of his 3-ft. apron, if he had found it necessary to force his shoe and lower sets downward separately? If he had to begin this method of sinking, he would have to continue it till the shaft was completed.

# STORAGE BATTERY LOCOMOTIVES FOR MINE SERVICE.

By T. H. Crosby.

Annual Meeting, British Columbian Division, Vancouver, February, 1922.

There are various sources of motive power available by which to apply energy through tractive effort. Some that come to mind that are available, or might be made available, for hauling cars in mines, are men and animals, compressed air locomotives, the so-called gasoline locomotive, and perhaps, under special conditions, the steam locomotive. Then we have the well-known electric trolley locomotive. In addition to these we have available to-day, as standard equipment, the storage battery locomotive, which is the latest addition, so to speak, and one which, in a large number of cases, affords the most economical and satisfactory means of hauling the material at hand. We are here only considering the moving of materials in cars or trucks on tracks with moderate grades, and are not including the service in which hoists or elevators are used, where the materials are usually, though not always, moved vertically or on inclines approaching the vertical.

Electrical mining locomotives have to-day a field of operation, in gathering and haulage service, in which they have no really serious competitor. In this field the storage battery locomotive is peculiarly adapted to gathering service and light haulage, leaving the main haulage work in the majority of cases to the trolley locomotive. The decision in any particular case, however, as to whether it is best to install a trolley locomotive or a storage battery locomotive depends, not on whether the work to be done can be classed as gathering or main haulage service, but upon the conditions surrounding the case in point, such as length of trip, number and weight of cars and load, average and maximum grades, possibility or desirability of installing a trolley wire and maintaining it as well as the necessary rail bonds, etc.

Some advantages of the storage battery locomotive may be enumerated as follows:

(1) It is for the time being, and within the limit of the battery charge, a self-contained power unit and is available for emergency service at all times.

(2) Elimination of trolley wires (and rail bonds) in places where the head room or other conditions prohibit their use or at least make them undesirable.

(3) Reduced fire risk.

(4) Energy for charging the battery may be drawn from the service mains during off-peak hours, and at a rate much less than the maximum rate at which energy is used by the locomotive when in operation.

(5) Great overload capacity, especially for a sustained period.

(6) In the poorly ventilated portion of gassy mines, sparks from the trolley or wheels cannot be tolerated. The storage battery locomotive can be, and now is, made gas-proof when necessary, and will pass the tests required by the U.S. Bureau of Mines for locomotives for operation under such conditions.

(7) In a number of cases, trolley locomotives would not be installed in place of animal haulage, but storag battery locomotives would show extremely attractive savings.

The storage battery locomotive must not be expected, however, to economically handle all classes of haulage, nor to equal the performance, nor have the service capacity, of a trolley type locomotive of equal weight or even lesser weight, because it has to haul its own source of energy, namely, the storage battery. The battery capacity is limited by the available space and permissible weight; also, a considerable period of time is required in which to re-charge it, after a previous charge has been exhausted. However, in intermittent service, where the hauls are not unreasonably long and grade conditions not very severe, the storage battery locomotive can be operated to better advantage, and more cheaply, than any other form of haulage apparatus.

The relative cheapness of haulage by storage battery locomotive is illustrated in the following table, which gives for comparison the costs of storage battery and animal haulage on service where six mules are replaced by one locomotive:

| | |
|---|---|
| Cost of locomotive, charging panel, etc | $8,000.00 |
| Interest at 8% | 600.00 |
| Depreciation on loco. chassis, 7½% on $4,000 | 300.00 |
| Repairs at $4.00 per month | 48.00 |
| Depreciation on battery per year | 800.00 |
| Labour — 1 hour per day at 75 cents per hour | 225.00 |
| One motorman at $7.00 per day | 2,100.00 |
| One trip-rider at $6.00 per day | 1,800.00 |
| Current for charging, say 30 K.W.hrs. per day at $1.50 | 450.00 |
| Cost of operating per year of 300 days | $6,323.00 |
| Cost of 6 mules at $125.00 each | $ 750.00 |
| Interest at 8% | 60.00 |
| Depreciation, 20% | 150.00 |
| Keeping mules, 365 days at $1.25 each, including feed, harness, shoeing, etc | 2,737.00 |
| Six drivers at $6.00 per day | 10,800.50 |
| Cost of operating 6 mules | $13,747.50 |
| Cost of operating locomotive | 6,323.00 |
| Saving in operating storage battery loco. or over 90% of the investment saved per year. | $7,424.50 |

If the locomotive be compared to man haulage, with the above conditions, at least ten or twelve men would be replaced by the locomotive and its crew of two men. We then have:

| | |
|---|---|
| 10 men at $6.00 per day for 300 days | $18,000.00 |
| Cost of operating locomotive as above | 6,323.00 |
| Saving by locomotive operation or over 140% of the investment saved per year. | $11,677.00 |

Cases are on record where the introduction of storage battery locomotives has reduced the cost of haulage from 45 cents per ton to less than 15 cents per ton. In some of our local mines the saving has been of the same order.

In comparing the relative cost of operating storage battery locomotives and trolley locomotives, one or other will be the more attractive, depending on the individual sets of conditions in the particular case in hand.

It is only seldom that the storage battery locomotive can be adapted to main haulage service in coal mines, on account of the requirements of such haulage being, in general, too severe. The hauls are too long, the grades are usually too great, and the schedule speed required to maintain the mine output is too high. Their principal field in a coal mine is in gathering service, in which they can advantageously displace animal haulage and, in many cases, the cable reel or traction

reel locomotive. The storage battery locomotive, when properly installed, has proved itself to be an extremely attractive investment. In gathering service the hauls are, in general, comparatively short and grades, even if severe, are short. When operating on these grades the locomotive is required to haul, as a rule, only one or two cars, which is not prohibitive.

In metal mines, however, these locomotives can usually be employed for main haulage, as the conditions are more suitable for their operation; the hauls are comparatively short and the grade is usually nearly level and slightly in favour of the loaded portion of the trip.

In most installations, the battery can be given boosting charges at high rates during periods of idleness, which results in increased capacity. With a properly arranged cycle of operation, the locomotive can be kept in service continually, 24 hours per day, by means of boosting charges only. The Ironclad Exide battery is especially suited to this service, as it can be boosted efficiently, can be discharged at high rates, and is capable of two complete discharges per day.

Storage battery locomotives should not be expected to operate at speeds equal to those usual with trolley locomotives, because of the effect of high discharge rates on the battery. The Watt discharge rate of the battery, at any instant, is equivalent to the product of the Watt-hours per train mile and speed in miles per hour. The useful Watt-hours per train mile, that is, the Watt-hours per train mile at the locomotive wheels, are approximately equivalent to twice the tractive effort, since one Watt-hour is equivalent to 2,653 foot pounds. The actual Watt-hour discharge will be equivalent to the useful Watt-hours per train mile divided by the efficiency of the energy conversion from the battery to the rims of the wheels. The actual Watt-hour discharge per train mile for a specified tractive effort is, therefore, constant, so that the discharge rate in Watts from the battery will be directly proportional to the speed in miles per hour. Thus, the higher the speed, the harder the battery will be worked and the less will be the overall efficiency of the combination. It is accepted practice to design storage battery locomotives of the mine type to operate at 3½ miles per hour at 80 volts, when developing their rated draw bar pull. When operating from the battery, the speed is somewhat greater, since the average discharge

voltage of a 48-cell lead acid battery or an 80-cell Edison battery is greater than 80 volts. A storage battery locomotive is seldom called upon to sustain its rated draw bar pull in service, so that its average operating speed will be considerably higher than its rated speed. The average operating speed of a trolley locomotive is higher than that of a storage battery locomotive, but the latter runs at a consistent speed throughout the day with a lower percentage of idleness than a trolley locomotive, thereby being capable of more economical operation than if operated at the higher speed of the trolley locomotive.

The storage battery locomotive possesses the distinct advantage of flexibility of operation. It can be employed in any portion of a mine where animal haulage can be used, and the low height type can operate in low workings in which animal haulage cannot be used. With the use of these locomotives, there is freedom from trolley wires, feeders, and cables, while the advantages of the electrical locomotive are secured. The flexibility of the track system is a particularly desirable feature, and one that is being appreciated to a greater extent as the use of the storage battery locomotive increases.

In selecting a storage battery locomotive for a particular service, it is well to remember that the apparatus is to-day to a large extent standardized and the usual advantages of using standard equipment, if at all possible, fully apply. It is essential to have a locomotive that has sufficient total weight to exert the necessary draw bar pull on starting and on grades, under average, or poorer than average, conditions of track. This can be determined when the number and weight of the cars and the weight of the load is known, and when car friction in pounds per ton has been ascertained by experience or experiment. The motors ordinarily must be large enough to slip the wheels of the locomotives, and must be able to operate throughout the complete cycle without undue heating or arcing at the brushes.

The battery, being the temporary reservoir of energy, must be of sufficient size to meet the instantaneous maximum demands which the motors make on it, and to have an available capacity to work the shift without re-charging or materially interfering with advantageous or customary working hours.

With the data just mentioned at hand in a complete form, the most suitable sizes and capacities can readily be determined by comparatively simple calculations.

In order to charge the batteries, a source of direct current must be available. In the case of the small power plant this source may be the direct current generator or dynamo which is used for lighting. Again, a separate generator of suitable size might be installed if necessary.

In larger plants, where alternating current is used in the electric power system, a motor generator set is installed. The direct current generator is direct driven by an alternating current motor supplied with power from the alternating current system.

A suitable charging panel is required for each charging station, and an additional control panel for the generator, if the one generator supplies a number of stations. These, however, are small items of the equipment and entail little expense. Automatic battery charging can be arranged for if desired, and although this involves a somewhat greater initial cost, it does away with a large portion of the supervision during the charging period.

With regard to the battery itself, there are to-day two main types available: the lead battery, with positive and negative plates of oxide of lead, and the nickel-iron alkaline battery, usually known as the Edison battery. Either type will give entire satisfaction when in service, though there are wide differences between the two in construction and performance, and each has its strong advocates. It is not necessary here to debate their relative advantages. It is a fact that either the lead, the acid, or the alkaline battery will give satisfactory results when the battery is suited to the work to be done, and when it has been designed with sufficient ruggedness to withstand the severe mechanical shocks met with in underground service, such as switching, derailments, and even collisions. In general, however, the Edison battery is probably preferable in all cases where battery supervision is casual or nearly absent. In places where there are enough storage batteries working to necessitate a battery-man being retained, the lead battery may be recommended, owing to its higher electrical efficiency, and higher available discharge rate.

# DEPLETION.

By R. V. Norris.

Annual Western Meeting, Vancouver, November, 1922.

The Standard Dictionary defines "deplete" as follows: "To reduce or lessen, as by use, exhaustion, or waste; to empty or partially empty".

In the so-called wasting industries, especially exemplified by mining, depletion is a material factor which must be taken into account in properly determining earnings and valuations.

This is clearly stated by Hoover in his *Principles of Mining* (p. 42): "As every mine has a limited life, the capital invested in it must be redeemed during the life of the mine ..... in mines, a portion of the annual income must be considered as a return of capital, therefore, before the yield on a mine investment can be determined, a portion of the annual earnings must be set aside in such a manner that when the mine is exhausted the original investment will have been restored".

The restoration of capital may be accomplished (1) by setting aside yearly a portion of the earnings which, at compound interest, will amount to the original investment on the exhaustion of the property — this method is known as *amortization;* or (2) by a unit charge against the mineral, sufficient in the aggregate to equal, on exhaustion, its original value — such unit charge is known as *depletion.*

In amortization, the replacement of value is partial and is completed by compounding interest; in depletion, the entire value is at once replaced and the depletion allowance may be used to retire bond issues or in partial payments on stock during the life of the property.

The need in the wasting industries for a depletion allowance in the determination of earnings has been recognized in all the Income Tax laws of the United States. In the 1913 law this was crudely attempted by allowing for depletion an amount not exceeding 5% of the value of the product at the

point of production. This sometimes resulted in the levying of a tax on an actual loss, and was corrected in later laws to a "reasonable allowance".

The Revenue Act of 1921, the present law, reads as follows :

[Sec. 214. (a) "That in computing net income there shall be allowed as deductions:]

"(10) In the case of mines, oil and gas wells, other natural deposits, and timber, a reasonable allowance for depletion and for depreciation of improvements, according to the peculiar conditions in each case, based upon cost including cost of development not otherwise deducted: PROVIDED, That in the case of such properties acquired prior to March 1, 1913, the fair market value of the property (or the taxpayer's interest therein) on that date shall be taken in lieu of cost up to that date: PROVIDED FURTHER, That in the case of mines, oil and gas wells, discovered by the taxpayer, on or after March 1, 1913, and not acquired as the result of purchase of a proven tract or lease, where the fair market value of the property is materially disproportionate to the cost, the depletion allowance shall be based upon the fair market value of the property at the date of the discovery, or within thirty days thereafter: AND PROVIDED FURTHER, That such depletion allowance based on discovery value shall not exceed the net income, computed without allowance for depletion, from the property upon which the discovery is made, except where such net income so computed is less than the depletion allowance based on cost or fair market value as of March 1, 1913; such reasonable allowance in all the above cases to be made under rules and regulations to be prescribed by the Commissioner, with the approval of the Secretary. In the case of leases the deductions allowed by this paragraph shall be equitably apportioned between the lessor and lessee; * * *"

This is perfectly fair, and the difficulty is not with the principle, but with its application, particularly in securing the necessary data to determine the unit value for depletion, which is the value of the property divided by the number of units of mineral.

In explanation, the Treasury "Regulations 62" states [Art. 201]:

" ***** taxpayers shall be allowed as a deduction in computing net income a reasonable allowance for depletion of mineral and for depreciation of improvements, ***** the essence of these provisions of the statute is that the owner of mineral deposits whether freehold or leasehold, shall, within the limitations prescribed, secure through an aggregate of annual depletion and depreciation deductions the return of (a) the cost of his property if acquired subsequent to March 1, 1913, or (b) the value of his property on the basic date, plus subsequent allowable capital additions, but not including land values for purposes other than the extraction of minerals."

It should be noted that *depletion*, not *amortization*, is allowed. This is logical, as the ultimate value of a mining

property is reduced by each unit of mineral removed, not by some amount obtainable on the exhaustion of the property by placing certain sums yearly at compound interest.

The basic date, "March 1, 1913", is the date of the adoption of the 16th Amendment to the Constitution of the United States, which permits an income tax.

METHODS OF VALUATION.

The United States Treasury Department permits a wide latitude in determining the value of mineral properties, subject to approval or revision of the Commissioner, the test being the "fair market value" of a property, which is the amount which would induce a willing vendor to sell, and a willing buyer to purchase. Consideration is given to cost, actual sales of similar properties, market value of stock or shares, royalties or rentals, value fixed by owner for capital stock tax, valuations for local or State taxation, partnership accountings, records of litigation, Probate Court inventories, and, as stated, "in the absence of better evidence, disinterested appraisals by approved methods".

Present value methods are not considered to have great weight if the value of the deposit can be determined by cost or replacement value, if the presence of mineral has not greatly enhanced the value of the property, if the removal of the mineral does not greatly reduce the value of the property, and if the profits are wholly or in great part due to manufacturing or selling ability of the taxpayer.

Except, perhaps, under the above conditions, it is believed that present value methods represent the fairest method of appraisal for mining properties. This method has the endorsement of practically all engineering writers, headed by Hoover and Finlay, and it was adopted by the Taxation Committee of the American Institute of Mining and Metallurgical Engineers in 1919.

The situation has been ably summed up by Walter Renton Ingalls in his *Wealth and Income of the American People:* "The fundamental principle of mine valuation is the present worth of an annual dividend accruing during the period of years corresponding to the life of the mine".

Accepting as the proper value of a mining property the present value of future earnings, we have as principles of valuation:

(1) The total value of a property at any date is the value of the future earnings of such property discounted to that date.

(2) The value of the mineral in the ground at any date is the total value of the property as above, less the value of the present and prospective capital expenditures for plant, development, and equipment necessary to recover such mineral discounted to the same date.

To obtain this value the following factors must be determined:

(a) Tonnage available. This can be estimated with a great degree of accuracy for regular deposits of coal, iron ore, disseminated copper, and the like; for less regular deposits the best information available must be used.

With respect to reserves, only such mineral as can be mined within a reasonable time should be included as available, and all in excess of this should be classed as "reserves unavailable". This should be carried as a separate item and under no circumstances be subject to depletion. It is suggested that a proper time limit for available mineral shall be such that its present value on an annuity basis would be ninety per cent of the present value of an infinite amount, i.e., a perpetual annuity. At six per cent discount rate, this time limit would be forty years life, and at eight per cent, thirty years, which may be properly considered as maxima.

Properties with shorter lives would of course have depletion percentages calculated on the ratio of the present value for estimated life divided by years of life. The life of any property would be properly calculated on estimated average future output, taking into account probable future changes as well as past experience.

(b) Life of property. With careful study this can be estimated with reasonable accuracy, and even considerable errors in total life, if this be fairly long, have but small influence on the present value.

(c) Capital charges for development, plant, and equipment. The value of existing plant, etc., is readily ascertainable, and the value of future necessary capital expenditures can be reasonably estimated.

(d) Profit from operation. This is the undeterminable factor and can only be estimated to the best judgment of the valuing engineer.

In estimating the probable profit from operations, it is proper to consider probable future changes in output, or in value of mineral, either increases or decreases, and also to take into account variations in earnings due to changes in conditions or in output, and the trend of values as shown by past results. As an example of this, a study of the royalties paid in the anthracite region of Pennsylvania resulted in demonstrating an average advance of almost exactly half a cent per year, as shown on the diagram below.

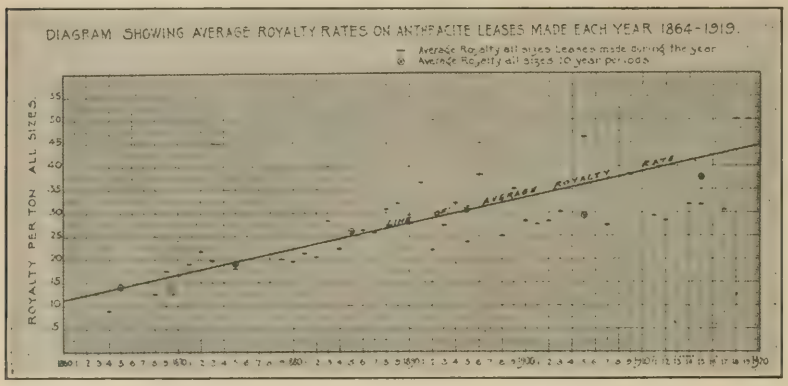


DIAGRAM SHOWING AVERAGE ROYALTY RATES ON ANTHRACITE LEASES MADE EACH YEAR 1864-1919.
Average Royalty all sizes Leases made during the year
Average Royalty all sizes 10 year periods
ROYALTY PER TON ALL SIZES.
AVERAGE ROYALTY RATE

## CALCULATION OF VALUATION.

Having determined the above factors, the valuation is arrived at by discounting to the desired date, at compound interest,

(1) The total profit estimated for each year, by deducting estimated cost from estimated selling price, including in cost all operating expenses, depreciation, and depletion, general and selling expenses, local taxes, etc., but not including interest charges, Federal income taxes, additions to surplus, dividends, or capital expenditures from income; and

(2) The estimated capital expenditures for plant, etc., year by year; to this value, discounted to the desired date, should be added the value of plant, capital development, and equipment existing at the date of valuation.

The results from (1) will give the total present value of the property; from (2) the present value of present and future equipment, etc., for depreciation; and (1) minus (2) will give the value of the mineral for depletion. This divided by the available tonnage [(a) above] gives the unit value for depletion.

## DISCOUNT RATE.

The question of the proper discount rate to be used to obtain present value is a mooted one; and, under various conditions, rates from 10%, or even more, down to 4% and under, have been used by competent authorities, while the need of higher percentages is recognized in doubtful cases, as in mines where the amount of mineral remaining is not ascertainable. In the case of mines in well-known territory, a 6% discount rate is considered proper. This is the legal rate of interest, and while it may be contended that a business like mining should yield a higher rate of profit, the experience of years shows that the average profit in most mining ventures is below rather than above this percentage.

It seems that, at least for taxation purposes, the 6% figure should be used. This has the authority of a standard legal rate, and anything else, except to discount uncertainties, must be a mere assumption.

While 6% is believed to be proper in most cases, it is recognized that a higher discount rate may be justly used where the extent or continuity of the deposit is doubtful.

## DEPLETION.

This amount, taken on all available tonnage estimated, will return the value of the property on the exhaustion of the estimated tonnage. The method has the advantage of a known and fixed tonnage depletion, but it has also the grave disadvantage that year by year the depletion has no standard relation to earnings. In bad years or cycles the earnings may be actually less than the depletion allowance. The result of this method is that the depletion allowance adds unduly to the cost of mineral in years of low output and excessive cost, and is negligible in years of high output and resulting low cost, so that the taxes paid are unduly irregular, and in poor years there may easily be no remaining earnings for taxation.

## SUGGESTED METHOD FOR DEPLETION.

It is suggested that the tax be based on earnings in lieu of tonnage depletion. The procedure outlined above would be followed in principle, but, instead of the figures being based on estimated earnings, the actual earnings for each year would be used, and the depletion be calculated as a percentage on earnings, based on the ratio of the present value of an annuity of $1.00 for the estimated life of the property, divided by the years of estimated life. This method gives practically the same ultimate result as the tonnage depletion, except that it substitutes in the calculation the actual for the estimated earnings, and varies the depletion year by year in relation to earnings. It, however, returns to the owner the actual value of his property at the basic date, based on the value of its earnings, instead of an estimated value. For the Government, this method yields taxes in all years where there are any earnings, and equalizes the tax returns.

If it is objected that this method might result in excessive depletion for the war years, it could be modified by using, instead of the actual earnings for those years, the average earnings of the pre-war years, say from 1910 to 1916 inclusive; or the objection is probably removed by not making the ruling retroactive, as the earnings of the natural resources industries have now returned to practically normal.

The percentages of earnings taken for depletion for a life of the operation from one year to the limit of calculation at 6% and 8% interest rate are shown in the table.

This agrees with the usual calculation, as is shown by the following example: Assume a coal property with very large reserves, so that the limit of forty years' life would be exceeded, and estimated to produce 1,000,000 tons annually at a profit of 25 cents per ton.

| | |
|---|---|
| Life | 40 years. |
| Tonnage available | 40,000,000 tons. |
| Average annual profit | $250,000. |
| Discount rate | 6%. |
| Present value $1.00 annuity | $15,046. |
| Present value available tonnage | $3,761,500. |
| Depletion per ton | 9.404 cents. |
| Depletion of average earnings | 37.61%. |

| Year of Life. | 6% INTEREST Per cent earnings allowed for depletion. | 8% INTEREST Per cent earnings allowed for depletion. |
|---|---|---|
| 1 | 94.34 | 92.59 |
| 2 | 91.67 | 89.16 |
| 3 | 89.10 | 85.90 |
| 4 | 86.63 | 82.80 |
| 5 | 84.25 | 79.85 |
| 6 | 81.96 | 77.09 |
| 7 | 79.75 | 74.37 |
| 8 | 77.62 | 71.83 |
| 9 | 75.57 | 69.40 |
| 10 | 73.60 | 67.10 |
| 11 | 71.69 | 64.89 |
| 12 | 69.87 | 62.80 |
| 13 | 68.09 | 60.79 |
| 14 | 66.38 | 58.88 |
| 15 | 64.75 | 57.06 |
| 16 | 63.16 | 55.32 |
| 17 | 61.63 | 53.64 |
| 18 | 60.15 | 52.06 |
| 19 | 58.72 | 50.54 |
| 20 | 57.34 | 49.09 |
| 21 | 56.01 | 47.70 |
| 22 | 54.73 | 46.36 |
| 23 | 53.49 | 45.09 |
| 24 | 52.29 | 43.87 |
| 25 | 51.13 | 42.70 |
| 26 | 50.01 | 41.57 |
| 27 | 48.93 | 40.49 |
| 28 | 47.87 | 39.47 |
| 29 | 46.86 | 38.47 |
| 30 | 45.88 | 37.52 |
| 31 | 44.93 | 36.61 |
| 32 | 44.12 | 35.73 |
| 33 | 43.12 | 34.89 |
| 34 | 42.26 | 34.03 |
| 35 | 41.10 | 33.30 |
| 36 | 40.61 | 32.55 |
| 37 | 39.82 | 31.82 |
| 38 | 39.05 | 31.13 |
| 39 | 38.33 | 30.46 |
| 40 | 37.61 | 29.81 |

With the proposed plan the calculations would be simplified, the most serious question, that of probable profit, would be eliminated, and the division between lessor and lessee becomes absolutely simple, as the royalty and profit (over royalty) would be handled separately on the same percentage. The one doubtful question would be the life of the property, and this is of relatively minor importance after the first years.

## ACCOUNTING.

Any proper depletion charge is dependent upon accurate and proper accounting. To those needing a model, the Standard System of Accounting, adopted by the National Coal Association (Commercial Bank Bldg., Washington, D.C.), is heartily recommended as simple and ample for ordinary conditions.

It is of very great importance that accounting be uniform, and the adoption of a standard system for each branch of the industry is almost imperative.

In this connection the latest Regulations (62) of the United States Treasury Department, read:

Art. 222. Allowable capital additions in case of mines.—

"(a) All expenditures for development, rent, and royalty in excess of net receipts, from minerals sold shall be charged to capital account recoverable through depletion, while the mine is in the development stage. Expenditures made in order to maintain the mine at its normal output shall be deducted as an expense in the year in which the expenditure is made or accrues. Any expenditure for extraordinary development and equipment, such as stripping, shaft sinking, tunneling, and other work beyond that necessary to maintain the mine at its normal production or output, should be carried forward and apportioned and deducted as an operating expense in the years to which it is applicable.

"(b) All expenditures for plant and equipment shall be charged to capital account recoverable through depreciation, while the mine is in the development stage. Thereafter the cost of major items of plant and equipment shall be capitalized, but the cost of minor items of equipment and plant, necessary to maintain the normal output, and the cost of replacement may be charged to current expense of operation."

It is somewhat difficult to properly draw the line between capital charges and operating costs as indicated above. The clause "necessary to maintain a normal output" indicates that the line may be drawn about as suggested below.

*Capital Charges.* — There should be charged to capital account:

(a) The value of the mineral in the ground;

(b) The value of the development;

(c) The value of mechanical equipment inside and outside; and

(d) The value of plant, buildings, dwellings, water-works, sewerage, roads, railroads, plant facilities, and the like.

All to *attain*, but not to *maintain*, output.

*Operating Charges.* — There should be charged to operation all costs of production, all development, and plant and equipment necessary to maintain output; also all overhead expenses necessary in carrying on the business, all local taxes, depletion, and depreciation.

*Development.* — The cost of the development of a property to its intended output is properly chargeable to capital. All further development to maintain output is properly operating expense. A mine may be considered developed when there are sufficient working places to produce the designed output, and sufficient entry work in progress to replace exhausted areas and maintain output. Sufficient advance development should be maintained to assure beyond question the maintenance of output under unfavourable conditions.

*Plant and Equipment.* — All plant and equipment necessary to bring the property to capacity is properly a capital charge. All additions, renewals, and extensions necessary to maintain output should be charged to operation.

In general, a rather liberal allowance for depletion is advisable to allow for possible, and even probable, lower unit profit in the later years of a mine's life, and for unforseen changes in conditions which may unfavourably affect the output.

The calculation of depletion, which is in effect a valuation of the mineral, requires high technical skill, careful work, and intelligent judgment, as a mere calculation of reserves with a crudely estimated profit may readily lead to grave errors.

# BALL PATHS IN TUBE MILLS

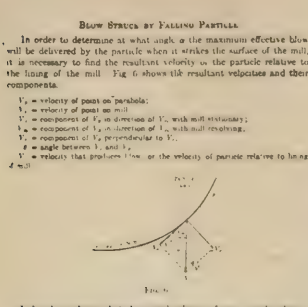
By H. E. T. Haultain and F. C. Dyer

Annual General Meeting, Ottawa, March, 1922.

There has been much written on ball mills, but no small amount of the literature is simply the expression of individual opinion without sufficient data. It may well be described, in the words of Mr. D. L. H. Forbes, as creeds rather than philosophy. This is no doubt due to the complexity and obscurity of the phenomena involved. The outstanding paper is "Fine Crushing in Ball Mills", by E. W. Davis, Superintendent, School of Mines Experiment Station, University of Minnesota, published in the *Transactions of the American Institute of Mining and Metallurgical Engineers*, Vol. LXI, 1920. This paper is based on a large amount of data from careful experiments with full sized apparatus, together with some beautiful mathematical philosophy. It contains much valuable and useful information and food for thought. An extract will show the spirit of it. After describing his large scale experiments and drawing conclusions therefrom, he writes:—

"In the endeavour to determine the best working conditions for the ball-mill, a detailed mathematical study was made of the action of the ball charge. While the data taken from a properly conducted test are convincing, an engineer sometimes prefers a mathematical proof. Test data contain a large personal factor, not only of manipulation but also of the person reporting the results. In the case of ball-mill crushing the amount of available data is enormous and, by careful selection, nearly any statement can be "proved". For this reason, a consideration which is entirely theoretical and devoid of any personal element would seem to be desirable and instructive."

Figure 1 shows two out of the many pages of math-



BLOW STRUCK BY FALLING PARTICLE

In order to determine at what angle $\alpha$ the maximum effective blow will be delivered by the particle when it strikes the surface of the mill, it is necessary to find the resultant velocity of the particle relative to the lining of the mill. Fig. 6 shows the resultant velocities and their components.

$V_p$ = velocity of point on parabola;
$V_m$ = velocity of point on mill;
$V_{p_n}$ = component of $V_p$ in direction of $V_m$ with mill stationary;
$V_{p_p}$ = component of $V_p$ in direction of $V_m$ with mill revolving;
$V_n$ = component of $V_p$ perpendicular to $V_m$;
$\theta$ = angle between $V_p$ and $V_m$;
$V$ = velocity that produces blow, or the velocity of particle relative to lining of mill.

Fig. 6.

It has been shown that the initial velocity of any particle when it started on its parabolic path is given by the equation $V = \sqrt{rg}\cos\alpha$. This is the velocity in the circular path of a particle of radius $r$. The velocity of any point on the parabola is given by the equation

$$V_p = \sqrt{Vg\cos^2\alpha + \left(V\sin\alpha - \frac{gx}{V\cos\alpha}\right)^2}$$

but $V = \sqrt{rg}\cos\alpha$ and $x = 4r\sin\alpha\cos^2\alpha$.

Then $V_p = \sqrt{9rg}\cos\alpha - 8rg\cos^2\alpha$.

The velocity of any point on the circle being $V_c = \sqrt{rg}\cos\alpha$

Then $V_p\cos\theta = V_m - V_{p_m}$, $V_p\sin\theta = V_{p_n}$; $V_n = V_m\sqrt{V_m^2 + V_p^2 - V}$.

Then $V_m = V_m\cos\theta - V_{p_m}$, and $V_n = \sqrt{(V_p\cos\theta - V_c)^2 + (V_p\sin\theta)^2}$

$$V_n = \sqrt{V_p^2\cos^2\theta - 2V_pV_c\cos\theta + V_c^2 + V_p^2\sin^2\theta}$$
$$= \sqrt{V_p^2 - 2V_pV_c\cos\theta + V_c^2}$$

$$V_n^2 = 9rg\cos\alpha - 8rg\cos^2\alpha + rg\cos\alpha - (2\sqrt{9r^2g^2\cos^2\alpha - 8r^2g^2\cos^3\alpha})\cos\theta$$
$$= rg\cos\alpha[10 - 8\cos^2\alpha - (2\sqrt{9 - 8\cos^2\alpha})\cos\theta].$$

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In order to determine the angle $\theta$ it is necessary to find the angle at which the circle and parabola intersect. This is done by computing the slopes of the tangents to both curves at this point and then the angle between these tangents. As the slope of the tangent is the first derivative of the equation of the curve, the slope of the tangent to the circle is

$$\frac{d}{dx}[x^2 + y^2 - (2r\sin\alpha)x + (2r\cos\alpha)y] = 0,$$

whence $\frac{dy}{dx} = \frac{2r\sin\alpha - 2x}{2y + 2r\cos\alpha}$ at the point $(x, y)$.

Then since $x = 4r\sin\alpha\cos^2\alpha$, and $y = -4r\sin^2\alpha\cos\alpha$,

$$\frac{dy}{dx} = \frac{4\sin^2\alpha - 3\sin\alpha}{4\cos^2\alpha - 3\cos\alpha},$$

which is the slope of the tangent to the circle.

Likewise for the parabola, the slope of the tangent is

$$\frac{dy}{dx} = \tan\alpha - \frac{2x}{2r\cos^2\alpha} \text{ at the point } (x, y);$$

$$\frac{dy}{dx} = -3\tan\alpha \text{ is the slope of the tangent to the parabola.}$$

The angle between two lines is expressed by the equation,

$$\tan\theta = \frac{m_1 - m_2}{1 - m_1m_2},$$

in which $m_1$ and $m_2$ are the slopes.

$$\tan\theta = \frac{-3\tan\alpha - \frac{4\sin^2\alpha - 3\sin\alpha}{4\cos^2\alpha - 3\cos\alpha}}{1 - 3\tan\alpha\left(\frac{4\sin^2\alpha - 3\sin\alpha}{4\cos^2\alpha - 3\cos\alpha}\right)}$$

$$= -\frac{8\sin^2\alpha - 4\sin^2\alpha + 1}{8\sin^2\alpha - 4\sin^2\alpha + 1}$$

But $\cos\theta = \frac{1}{\sqrt{1 + \tan^2\theta}}$

then $\cos\theta = \sqrt{\frac{64\sin^4\alpha - 64\sin^4\alpha - 8\sin^2\alpha + 1}{8\sin^2\alpha + 1}}$.

It is now possible to write the complete formula for the effective velocity of the particle.

$$V_n^2 = rg\cos\alpha\left[10 - 8\cos^2\alpha - (2\sqrt{9 - 8\cos^2\alpha})\sqrt{\frac{64\sin^4\alpha - 64\sin^4\alpha + 8\sin^2\alpha + 1}{8\sin^2\alpha + 1}}\right]$$

This equation may be simplified to $V_n^2 = 16kr\cos\alpha\sin^2\alpha$. (8)

But $\cos\alpha = kr$, and $\sin\alpha = \sqrt{1 - k^2r^2}$.

Then $\sin^2\alpha = 1 - k^2r^2$, and $\sin^4 = 1 - 2k^2r^2 + k^4r^4$.

and $V_n^2 = 16k^2r^2 - 32k^4r^4 + 16k^6r^6$

**Figure 1**

ematics. Figure 2 shows the visualization of some of the mathematics. Certain fundamental equations are developed as follows:—

But $k = 1.226n^2$, whence
$$K = \sqrt{\frac{0.443}{r_1^2n^4} - 1} \qquad (11)$$

when $r$ is the radius of the mill and $n$ is its speed in rev. per second.

The value of $K$ given by this equation is the relation between $r_1$ and $r_2$ which should exist for best operating conditions and is really the measure of the quantity of the charge. This equation then gives the proper relation between the size of the charge, the size of the mill and its speed. It is the fundamental equation of the ball-mill and shows the relations mentioned on page 264.

Equation (11) may be restated in the following forms, which may be more convenient.

$$n^4 = \frac{0.443}{(1 + K^2)r_1^2}$$

Then
$$n = \frac{0.8158}{\sqrt{r_1}\sqrt[4]{1 + K^2}} \text{ in rev. per sec.,} \qquad (12)$$

or
$$N = \frac{48.948}{\sqrt{r_1}\sqrt[4]{1 + K^2}} \text{ in rev. per min.} \qquad (13)$$

(16) $$\cos\alpha_2 = \frac{0.8165K}{\sqrt{1 + K^2}} = K\cos\alpha_1 \quad (19)\ C_a \text{ average} = 1.444$$

(20) $$K = -0.024 + 0.39\sqrt{7 - 10P} \text{ (very nearly)}$$

(22) $$\text{Hp.} = Wr_1^{3/2}\left[0.004467\frac{(1 - K^2)}{(1 + K^2)^{1/6}} - 0.0037\frac{(1 - K^3)}{(1 + K^2)^{7/6}} + 0.00088\frac{(1 - K^7)}{(1 + K^2)^{11/6}}\right]$$

By use of these equations, with any given set of conditions, it is possible to determine the geometrical shape of the charge in the mill, the horsepower absorbed by it, the velocity of the blow struck by any ball, and the number of blows struck by any ball. The equations show also the relation that should exist between speed, diameter of mill and size of charge in order to secure the maximum theoretical efficiency.

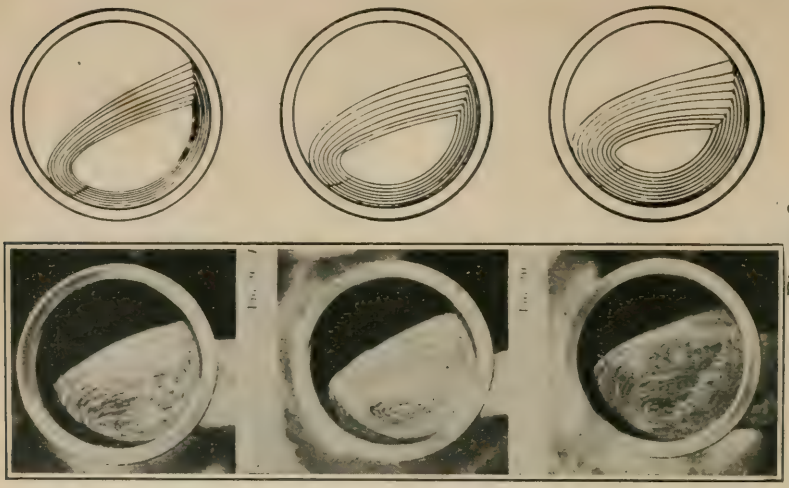


Figure 3

—PATHS OF TRAVEL OF PARTICLES IN AN 8-FT. MILL MAKING 24

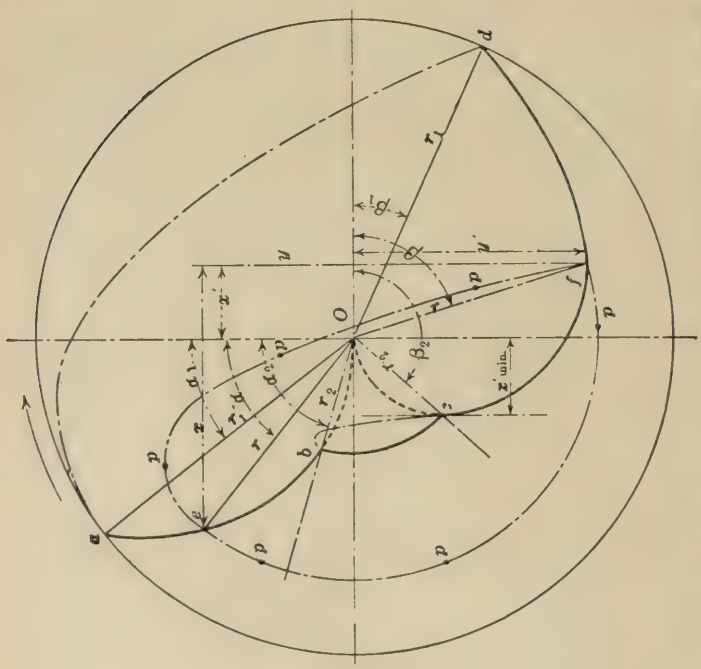



Figure 2

This is the most remarkable statement in the literature of ball milling. The whole philosophy is summed up in these equations. It is magnificent if true, but the present writers question the foundations upon which this philosophy is based.

Figure 3, taken from the Davis paper, is apparently a verification of the truth of that part of the mathematics dealing with ball paths. Photographs of curves of actually falling particles in a mill (3 inches in diameter) are apparently identical with the calculated curves. Spurred on by this work of Davis, the writers of this present paper continued the desultory work they had been doing along this line. At first, photographs were taken of the ends of the rods in a small laboratory rod mill. The mill was tilted very slightly towards the feed end, and the discharge end was removed. The rods remained in the mill long enough to permit of good observation and good photography.

At a later period a special mill was constructed (Figure 4), with a diameter of 24 inches inside the stepped lining. It consists of two plate-glass plates half an inch apart in a cast iron

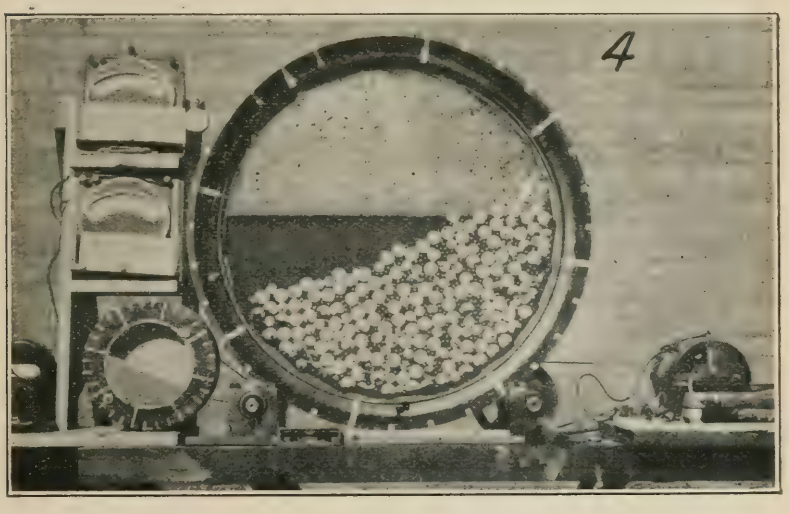



**Figure 4**

ring. It was driven by a variable speed motor and connected to an accurate speedometer (magneto and voltmeter type). In place of balls or rods, cast brass discs were used. These were of different diameters, from 2 inches down to three-quarters of an inch, and were three-eights of an inch thick. Four

small projections on each side prevented the discs sticking to the glass when used with water. Sometimes steel balls three-eighths of an inch in diameter were added to the load of discs. Figure 4 shows the lay-out as arranged for the movie. The load consists of water, with three-quarter inch discs painted white with a black radial line, three-eighth inch balls, and some specially prepared rubber with a specific gravity of 3.8. This latter was cut up into small cubes, about 10 to 20 mesh size, to represent ore. There is also a six-inch mill with corrugated linoleum lining and containing a dry ball load of mixed sizes. The mills are connected to the motor so that the small mill makes two revolutions to one of the larger mill.

Many runs were made, with a great variety of materials, including crushed marble, seeds of many kinds, and many mixtures, some dry and some wet. Figure 5 shows

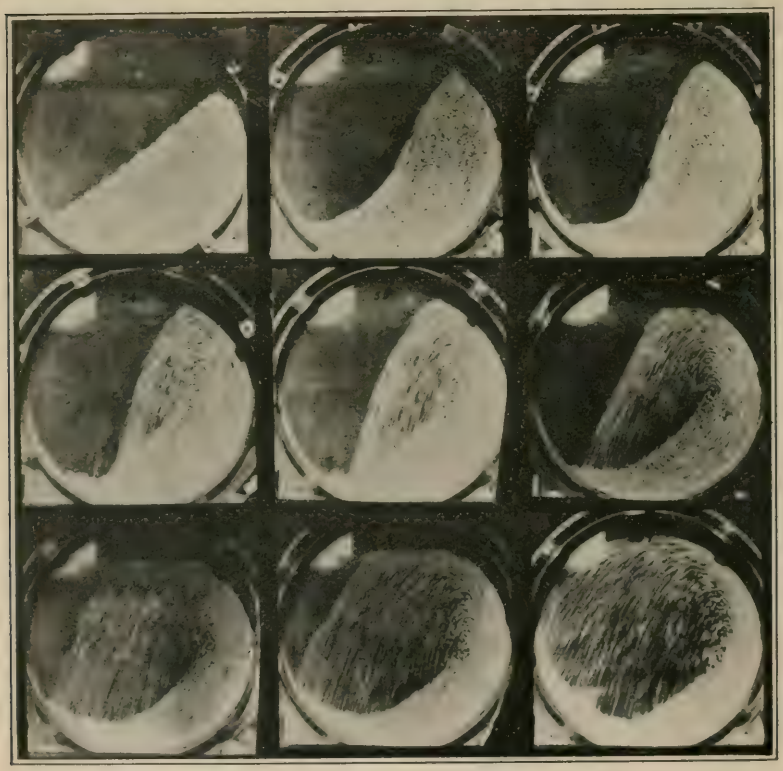


Figure 5

a series of photographs of small, hard, white peas. Nos. 56 and 57 check up with the Davis effect at the Davis speed, but the writers failed to get any such effects with discs or balls at

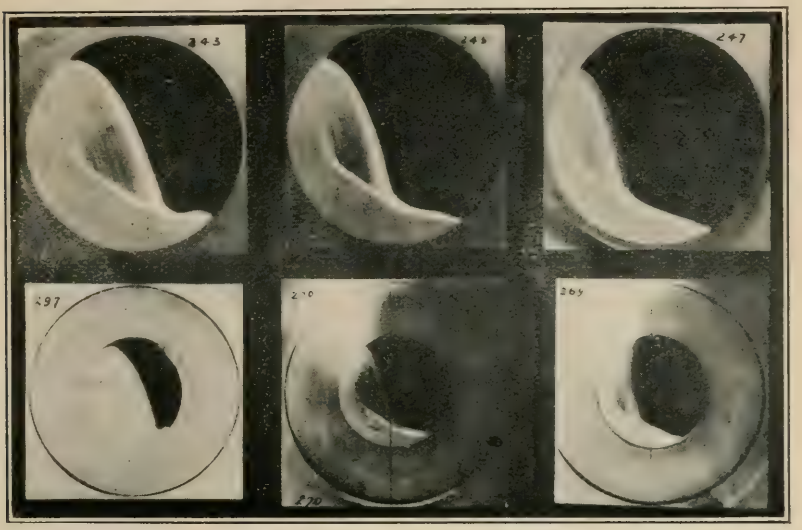


Figure 6

these speeds. Experiments were then conducted with mills six inches and three inches in diameter. Figure 6 shows results with small mills at the Davis speeds, in an effort

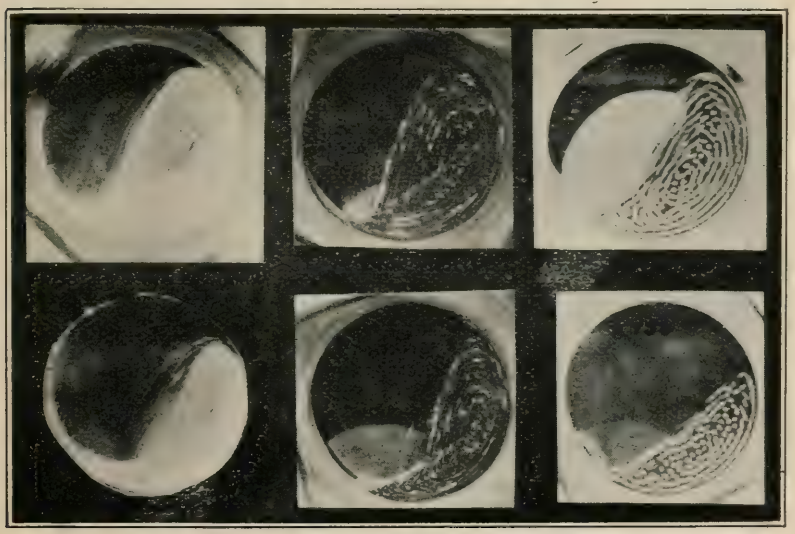


Figure 7

to get the Davis effect. The mills are of wood, with the surface roughened by coarse sand paper. The material is crushed marble, about 30 mesh with the fines screened out. A great many experiments were made, but the Davis effect did not arrive with the Davis speed. Figure 7 shows some more results with sand, galena, and shot. Note that the galena and the sand gave identical curves, and note also that the R.P.M. varies inversely as the square root of the diameter of the mill. Figure 8 shows the six-inch mill with 0.4, 0.3 and 0.2 loads, the upper series at the Davis speeds and the

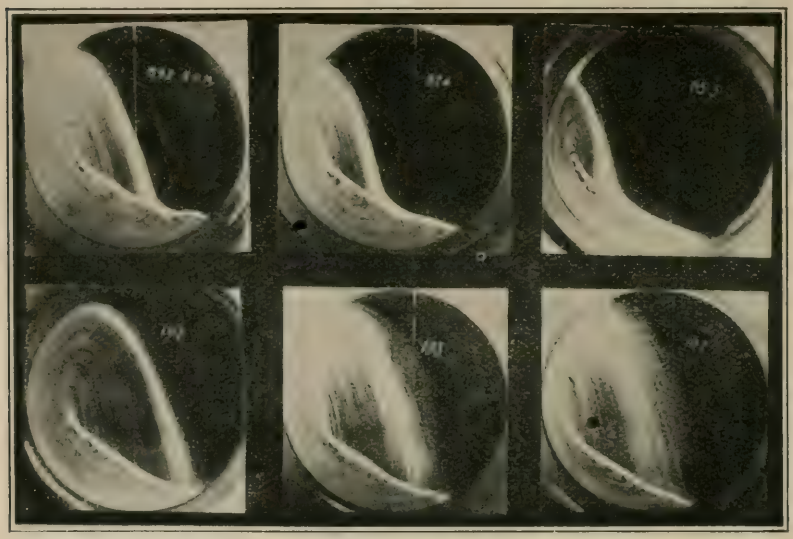


Figure 8

lower series with the Davis effect but at speeds considerably higher. Figure 9 shows three experiments, each at the Davis speed, in the six-inch mill, the centre one showing

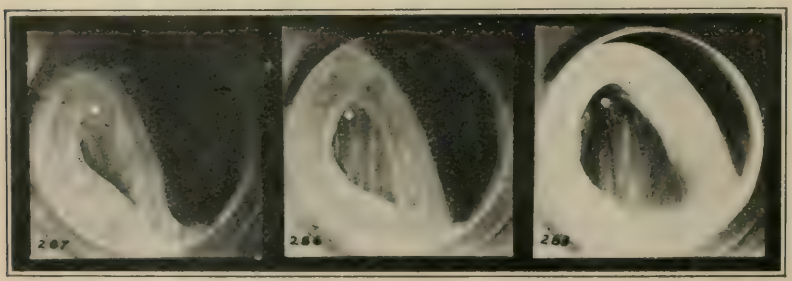


Figure 9

Figures 10, 14 and 18

the Davis effect. The photo to the left shows 60-mesh, the central one 120-mesh, and the right hand one 200-mesh, marlbe. Evidently the path of the grain depends, among other things,

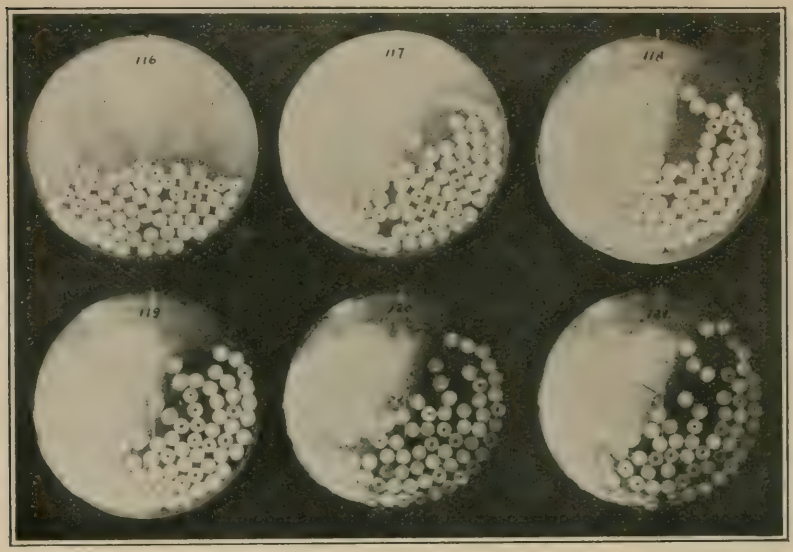



Figure 11

upon the character of the material. The character of the path

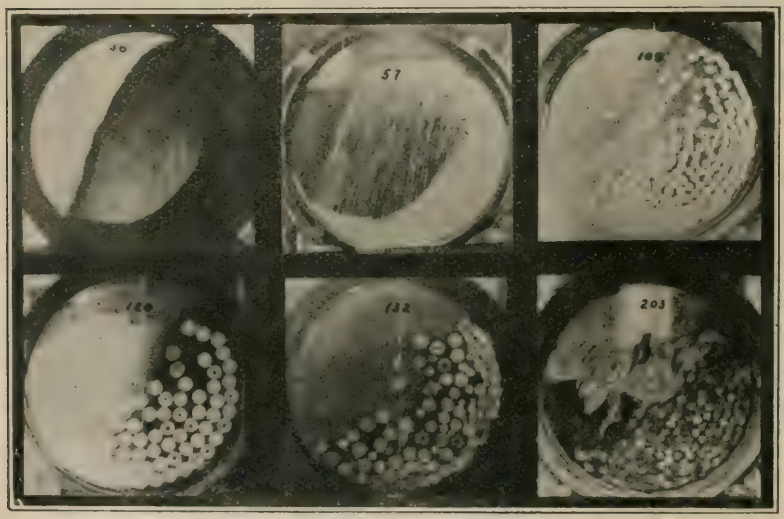



Figure 12

depends upon the amount of slip. Figure 10 shows a double effect. The six-inch mill was lined with corrugated linoleum and run at the Davis speed with 60-mesh marble. The grains carried up by the corrugations travel the Davis path for that R.P.M., the others do not.

Figure 11 shows an extreme effect of slip, where two-inch discs were used dry in the two-foot mill. Photo No. 120 is at the Davis speed, No. 121 is considerably faster. Figure 12 shows the two-foot mill at the Davis speed with different materials — rape seed, peas, one-inch discs, two-inch discs, mixed discs, one-inch discs in water.

Through the courtesy of Mr. Clifford Sifton and Filmcraft Ltd., a movie film was taken of the two-foot mill in action, using an ultra rapid camera at the rate of 120 photos per second. These are shown on the screen at the standard rate of sixteen

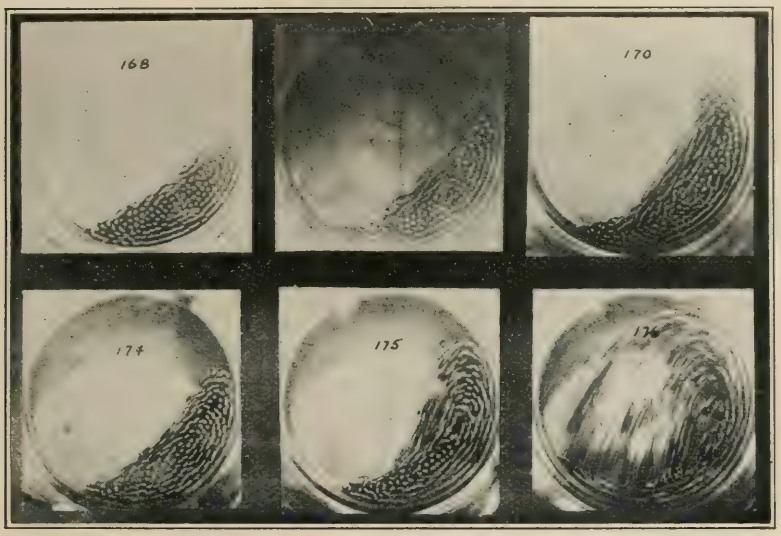



**Figure 13**

per second, thus slowing up the speed to about one-seventh. This film shows very plainly the slippage, or adjustment of position, as between the discs and the lining, and as between different layers of discs as they rise in the circular path. The black radial line enables one to see clearly the rotation of the discs relatively to each other in this adjusting of position. The film also shows to some extent the effect of segregation, and shows how, at some speeds, the hollows of the steps or waves

in the lining may be filled with particles of ore. It shows the effect of eddies in the pulp. It shows surges in the movement of the discs, these surges being apparently due to irregular slippage. It shows the great activity at the toe of the cascade, where reversal of direction takes place. It shows much more of the free fall parabolic paths with the balls in the small mill than with the discs in the larger mill.

These pictures are a beautiful aid to the visualization of what may take place in a ball mill, but they are not a perfect or complete representation of what happens inside either a ball mill or a rod mill. The discs are neither rods not balls. They do not have so many points of contact on each other as balls do, and consequently they may be more ready to slip. They do not carry up the ore and water as do rods, because the ore and water slip down between the discs and glass sides.

Despite this high speed movie, and the hundreds of photographs and thousands of visual observations, actual milling conditions have not been truly seen. We can only reason by analogy and by comparison. The movie shows conclusively the large amount of slip with discs, more slip with larger discs than with small. The sand photos show the effect of slip between the grains or layers of sand, apart from lining slip. In none of the experiments do we get away from slip, except with certain selected fine grain material, so fine that the particles adhere rather than slide. When shot was used with the roughened, but not stepped or corrugated, lining, the slip was evidently very great, but when a corrugated lining was used, a lining that would definitely lift a layer of shot, the Davis effect is approached at the Davis speed. This is illustrated in Figures 8, 13, and 14.

Figure 14 at the Davis speed appears to confirm the Davis effect. The outer curve is the mathematical non-slip curve as in Figure 2. If there were no slip between layers, the other balls would follow close behind and there would be a hole in the centre as in the Davis diagram and in Figure 9. Davis explains the absence of the hole in his sand pictures as due to the effect of *interference*. With the shot in Figure 14, it is clearly due to slip. Considering layer by layer of balls, the outer layer cannot slip on account of the

corrugations, but the next layer slips somewhat on the first layer. The slip between each succeeding pair of layers increases still more until we get to a neutral layer that is not sure whether it is rising or falling or stationary. No matter what the nature of the steps or corrugations, no matter whether there be only a light load in a six-inch mill or the heavy load in a six-foot mill, there must be slip or adjustment between balls in the circular rising paths. The line joining the centres of adjacent balls passes through an angle of more than 90 degrees in reference to the line of action of gravity. The corrugations *may* hold the outer layer of balls without slip, but there will be increasing slip with the other layers. Any slip or adjustment in the outer layer will be augmented in the others.

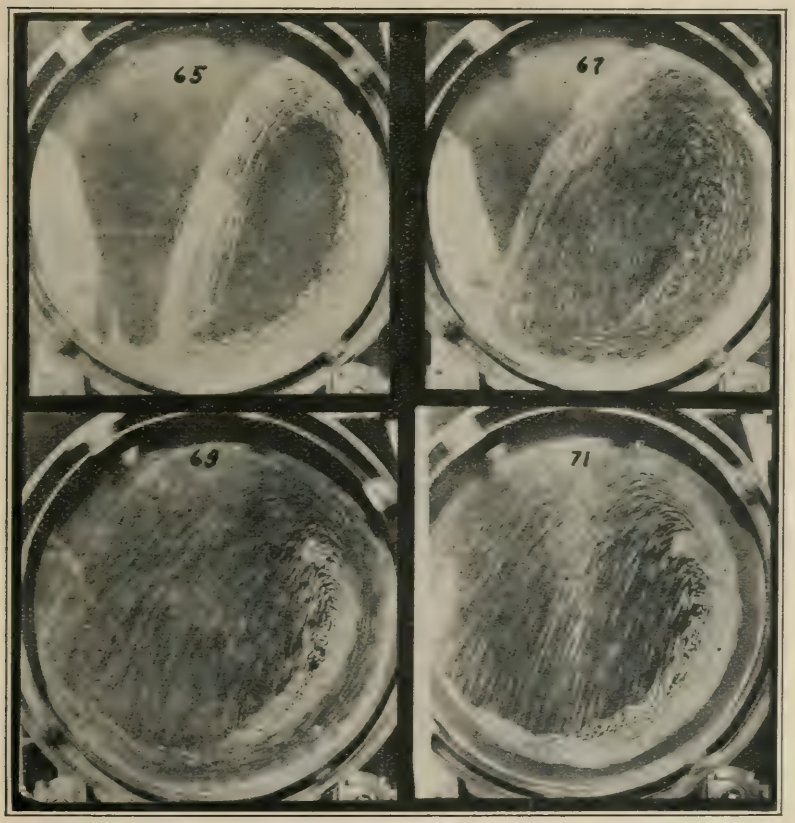



Figure 15

Of all the varying conditions existing in ball mills, slip is probably the least determinable, the most varying. The only sure thing about it is that it is always there[1].

In addition to the effect of slip on ball paths, segregation seems to be of importance. Figure 15 shows a mixture of black rape seed and white peas in the two-foot mill at different speeds. Note the small rape seeds in the centre at the lower speeds, and at the periphery at the highest speed. Figure 16 shows a mixture of 8-mesh quartz and 60-mesh hornblende behaving in the same way at different speeds[2].

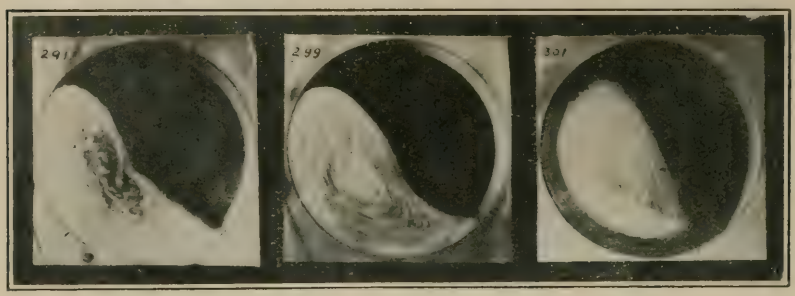



Figure 16

Figure 17 shows the six-inch mill at three different speeds, taken with a very short exposure[3]. Where there is much free parabolic fall (Davis effect), the large balls go to the centre where they get very little chance for work. The ore particles would tend to the periphery. (How important may be the thickness of pulp in this connection?)

The writers would urge a more definite understanding as to the use of terms applied to these ball paths, especially in regard to the word 'cascade', which is sometimes applied to the

(1) This paper is concerned with paths only, and does not consider the crushing effect of these paths. Work is progressing on the crushing phase of this problem. The writers cannot, however, refrain from expressing the view that slip may play a very large part in crushing. If one considers the rolling in the cascade zone as an exaggerated slip (as also at the toe of the cascade), then slip plays the dominant part in all crushing, except possibly in the case of coarsest crushing. As against this, Davis seems to base his whole philosophy on impact at the end of a long parabolic path, disregarding slip entirely.

(2) At the Annual Meeting a movie film was shown of this last experiment.

(3) The writers commend this illustration to the particular attention of Mr. Hardinge.

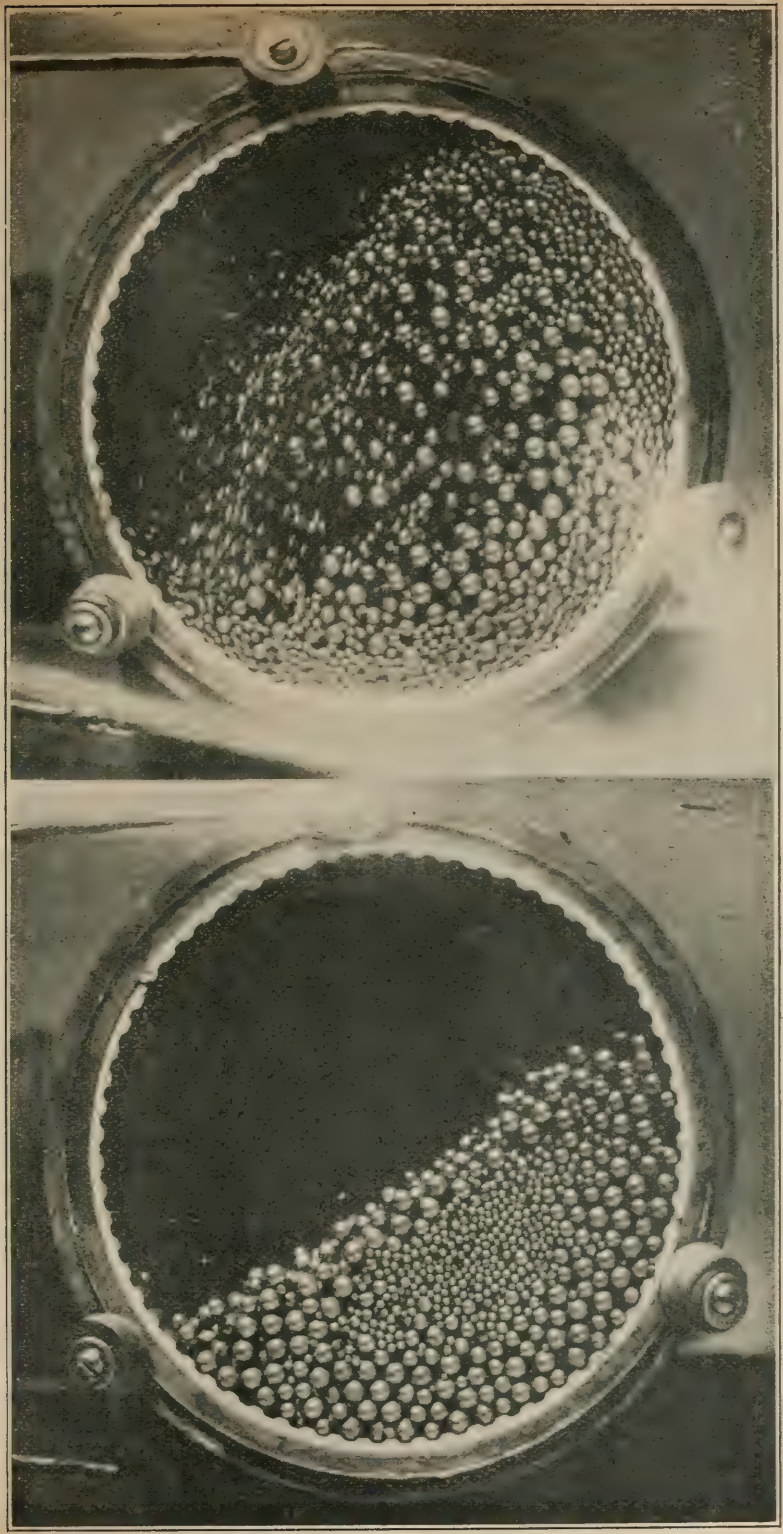


Figure 17

free parabolic path and sometimes to the balls rolling down over each other. There seem to be five or perhaps six definite zones in the ball paths, as shown in Figures 14 and 18. Zone No. 1 is the zone of the circular path. No. 2 is the zone of the parabolic path, or free fall, upon which Davis lays so much emphasis. No. 3 is the zone of cascade, shown more clearly in Figure 18. No. 4 is the zone of impact. No. 5 is the zone of reversal of direction. Between zones 1 and 3 there may be a neutral zone No. 6. Each one of these zones will have a larger or smaller part in the actual crushing, but this paper is not concerned with crushing effects.

In conslusion: This paper is a further illustration of the complexity of the subject. Their procedure and technique have features of novelty, but the writers do not claim to have produced any philosophy that has not been at least mentioned before. There have been so many suggestions and conjectures as to what goes on inside the mill that probably everything that can be imagined has been brought forward. The writers hope, however, that their work may help to clear up the ideas in the minds of some men, and thus prepare the way to further advance.

This paper represents a large amount of work done by the writers and their students[1]. About 500 photographs were taken in addition to the movie films. On the strength of this, the writers venture to express certain conclusions that they draw from their work:—

(1) Next to R.P.M., the most important factor in determining the paths of balls or rods in tube mills is slip.

(2) The amount of slip depends on several variables —the size of ball, the nature of the lining, the character of the pulp, and other factors — but slip is always present in practice.

(3) In some mills (in many mills), the effect of slip may be so great as to eliminate all free or parabolic fall at ordinary mill speeds.

(4) There is always a continuous change occurring in the relative positions of the balls in the upward or circular path.

(5) At the speeds adopted in practice there is more of the cascade effect than of the free fall or parabolic path effect.

(1) In connection with this, the names of the following students should be recorded:— T. C. S. Fawcett, B.A.Sc., B. T. Hannan, B. A.Sc., and E. A. Rolph, B.A.Sc.

(6) In tube mills, the paths of the balls and of the ore particles are affected by segregation. This seems to be due entirely to size and is but little affected by differences in specific gravity. The action is practically the same as that taking place in a talus or rock dump where the smaller pieces settle through the spaces between the larger, and the large pieces roll.

(7) At low speeds the small sizes are at the middle; at high speeds they tend to the periphery.

(8) The best mixing seems to be at the highest speed attainable without the parabolic path. If there is much of the parabolic path zone present there may be distinct segregation of the larger balls towards the centre and of the smaller balls and of the ore towards the periphery.

Note the difference between rod mills and ball mills in this respect. Segregation takes place between different size rods just as between balls, but the ore particles do not get the same chance to settle down through the rods as they do through the balls.

## THE DAVIS SPEED

*"R.P.M. for Best Theoretical Efficiency."*
*as given in his paper*
*"Fine Crushing in Ball Mills".*

| Inside Diameter of Mill | Portion of Mill Volume Occupied by Charge at Rest | | | |
|---|---|---|---|---|
| | 0.2 | 0.3 | 0.4 | 0.5 |
| 3 inches | 120.90 | 123.64 | 126.72 | 130.2 |
| 6 " | 85.50 | 87.44 | 89.62 | 92.08 |
| 1 foot | 60.45 | 61.82 | 63.36 | 65.1 |
| 2 feet | 42.75 | 43.72 | 44.81 | 46.04 |
| 3 " | 34.9 | 35.69 | 36.58 | 37.58 |
| 4 " | 30.22 | 30.91 | 31.67 | 32.55 |
| 5 " | 27.04 | 27.64 | 28.34 | 29.11 |
| 6 " | 24.69 | 25.24 | 25.88 | 26.58 |

## CONTRIBUTED REMARKS.

MR. E. W. DAVIS: I have read Messrs. Haultain and Dyer's paper through a number of times, and have gone over it with many people, in an endeavour to get a clear conception of the basis for the criticism of my paper on "Fine Crushing

in Ball Mills", which was published in the *Transactions* of the American Institute. As I understand it, there is no argument as to the mathematics involved or as to the balls following the mathematical paths under ideal conditions. The criticism seems to be based upon the claim that ideal conditions do not exist in ball mills and that, in fact, the discrepancy between actual and ideal conditions is very large. If the above statement is a correct summary of the ideas which Professors Haultain and Dyer wish to convey, I can find nothing in the paper sufficiently convincing to make me change my original views, as stated in the paper on "Fine Crushing in Ball Mills." It is apparent that, in the operation of the mill described by Professors Haultain and Dyer, slip largely determined the path of travel of the particles in the charge. The difference between the operation of their mill and the operation of grinding mills in practice is so great, however, as to cast grave doubt upon their conclusions. In other words, after studying the information in their paper, I can find no reason for changing my original opinion as stated in the *Transactions* of the American Institute.

Regarding the matter of slip, it may be said that considerable difficulty was met in the operation of the small mill, which was photographed for the illustrations accompanying my paper. It was very difficult to secure a grinding medium that could be photographed and which at the same time would not slip excessively in the mill. The inside of this small mill, which was made of steel, was knurled, and the charge used consisted of a mixture of sugar and hard coal, crushed to minus 48 mesh. The coal was used in order to produce the contrast needed in the photographs. In order to secure an accurate idea of the actual action of the charge, it was also necessary to keep the particles out of contact with the heads of the mill. The friction between the particles and the heads of the mill considerably deformed the natural curves. I do not believe that satisfactory results can be obtained with a mill such as that used by Professors Haultain and Dyer, in which the heads are only one-half inch apart. It would also seem to me that a large amount of slippage would hardly be expected in a large mill in which several tons of steel balls were carried by a corrugated steel lining and the whole mass well "lubricated" with a thick mixture of sand and water.

# CANADA'S COAL PROBLEM:

## A Further Study.

By F. W. Gray.

(Annual General Meeting, Montreal, March 1921.)

At the Annual Meeting a year ago, the writer read a paper on "Canada's Coal Supply", and at the Winnipeg meeting, in October, followed up this paper by adducing the later statistics of production and supply then available, and emphasizing the quite obvious importance of the western coalfield to Canada, arising from the concentration of 99.3 per cent of our coal reserves west of the Souris lignite beds.

In the events of the year that has intervened there has been much to encourage those who urge that the problem of Canada's coal supply should be viewed first from the standpoint of national independence and defence, and that *apparent* commercial standards should be relegated to their proper position of lesser importance. The word "apparent" is designedly chosen. The cost of coal to Canada has for some time been largely determined by the purchasing power of our paper dollar, and this is, in its turn, largely set by the ratio between the coal we produce and the coal we import; so that, paradoxical as it may seem, apparent cheapness from the individual viewpoint may be the rankest prodigality from a national viewpoint.

The domestic production of coal during 1920 was the largest in our history, and this is a matter for unreserved congratulation. As compared with 1919, we found 2½ million tons of coal, the equivalent of a $25,000,000 gold shipment to New York in support of national solvency.

Nevertheless, the 1920 production was 5 million tons below the production capacity of the existing coal mines — a quantity that, applied to domestic requirements, would have been equivalent to a further gold shipment to New York of not less

than $50,000,000, ignoring the stimulus to our internal economy that would have proceeded from the mining of this coal at home.

The output capacity of existing collieries is, however, by no means the capacity of domestic coalfields *situated within proved economically possible transportation distances of home markets.*

The maximum possible production of the existing coal mines of Nova Scotia and New Brunswick has, for the purposes of the foregoing calculations, been placed at 8 million tons per annum, but there is no insuperable difficulty in raising this production — given time and expenditure — to 10 million tons. Such a production would make the Maritime Provinces and Newfoundland, Quebec as a whole, and a considerable area of Ontario, completely independent of extra-national sources of supply. In making this statement, it is of course contended that anthracite is not a necessary import in any Canadian territory within economically possible transportation distance of domestic bituminous-coal mines.

The production of the western coalfield is anything that Canadian opinion and support likes to make it. There is no limit fixed by the availability of coal, either in regard to quality or quantity.

The gap — the apparently unfillable gap, under existing conditions of transport facilities — is Ontario. Even Ontario could be made accessible to Nova Scotian coal by a deep waterway from the sea to the Great Lakes, and in all probability will be so made accessible by a succeeding generation.

For the present, therefore, Ontario must look to the United States for coal; and, in recent years, Ontario could not look elsewhere for coal, seeing that even in those Canadian provinces plentifully supplied with coal, or situated within reach of the coalfields, the consumption exceeded the domestic production.

It is a proper thing that Ontario should import coal, because it is a necessary thing; but it is submitted that it is an improper thing from a national standpoint that the other provinces should import coal from extra-national sources, because it is unnecessary.

Two recent newspaper despatches may be quoted as instancing typical viewpoints that are based on error and are subversive of national interests.

One despatch, from Ottawa, dated January 18th, states: "That the danger of a coal famine in Canada this winter has been entirely removed, unless something should happen to interrupt importations of hard and soft coal from across the United States border, is the view expressed to-day by the Dominion authorities."

A second despatch, from Washington, quotes a U. S. Senator, speaking in a debate on a Bill that proposes the raising of a higher tariff wall against Canadian goods in the United States, as remarking: "Canada, as she has always done, will continue to buy where she can buy cheapest, and if she can buy cheapest in the United States, she will continue to buy from us."

The impression given by the Ottawa despatch is that it reflects a resignation in official circles to continued dependence upon the United States for coal, and this can only be based upon an unfortunate disbelief in the possibility of independence.

The jibe of the U. S. Senator galls deep, because it is true. That cheapness does not altogether consist in a relatively smaller outlay of money is not any less true.

These instances are quoted because they reflect a widespread opinion that it is the duty of the Institute to inform and correct.

The foregoing review about sums up the existing situation, which the writer submits is unsatisfactory in the following particulars:

(a) Coal output in Canada is disproportionately small in regard to the extent of the domestic coal reserves.

(b) The domestic output has not kept pace with the population of the country, and the growing consumption of coal per capita.

(c) That while some dependency on the United States for coal is inescapable, there is no good reason for the annually increasing dependency of recent years.

(d) That there does not yet exist a unified public opinion in Canada that will demand the use of domestic mined coal to the maximum point of substitution for imported coal.

Before considering possible remedies, the following facts regarding the relative coal resources of Canada are submitted, not as anything novel, but as bearing on the question at issue.

The coal beds of Canada, with a content estimated at 1¼ thousand billion tons, are about equal to those of the whole of Asia.

Canada has 17 per cent of the world's coal reserves, and 71 per cent of the reserves of the British Empire.

With a consumption per capita that will ultimately exceed that of less northerly countries, Canada has a production of 2 tons per capita; Britain and the United States have a production of 6 tons per capita.

The coal consumption of Canada has always been restricted by insufficient supply, nor have the combined domestic production and imports in recent years met the requirements of the country. Manufacturing has been restricted, selling prices have been unnecessarily high, and the country has never been on "easy street" in regard to coal supply. With present population, and potential manufacturing ability, Canada requires not less than 40 million tons of coal annually. The historical maximum production of 16 million tons in 1920 is entirely inadequate. With a population of 10 millions, Canada should use 50 million tons of coal annually, if she is to achieve maximum manufacturing ability, freedom of political action, and domestic comfort.

In regard to the attitude of the United States towards our coal supply, it may be suggested that, because of the exaggerated dependence of recent years that Canada has suffered through neglect to develop her own coal beds, we attach too great importance to what is, after all, a matter of quite insignificant value to the United States. That country has an annual coal production which exceeds 600 million tons, without having attained anything like the maximum possibility of production. The tonnage that the United States exports to Canada bulks small in that country's business, but to Canada it is a matter of economic life or death, and has probably the most important bearing on our continued political independ-

ence, being indeed our most pressing internal problem. We need not fear any criticism from informed opinion in the United States if this country develops its coal producing capacity to the utmost; but we do risk some justifiable and pitying criticism from the informed and well-disposed among our neighbours, and something worse from the ill-informed and hostile minority that is clamorous in inverse ratio to its numbers, if we continue to drop behind in the great international game of industrial emulation, in which coal and iron are the king pieces.

What are the remedies, and what of the future?

There are three territories to consider, namely:

(a) Eastern Canada, tributary to the Eastern Coalfield.
(b) The "Gap of Ontario."
(c) The provinces west of Fort William.

Two trade outlets are open for coal in the east — the home market (including Newfoundland), and the overseas export market. It does not seem an unreasonable aim that the Maritime Provinces and Quebec, including all the railways that serve this territory, should be supplied with domestic mined coal. The larger and more stable the market, the more cheaply will it be possible to mine coal in Nova Scotia, and it may be permissible to contend that on no national grounds can the purchase of coal from extra-national sources for national transportation lines be excused or defended without grave inconsistency.

The export trade, in view of the limited quantity of the coal reserves in Nova Scotia, is probably not one to be encouraged to a point of national deprivation; but it has never reached that point, and what promised to be the commencement of a permanent and profitable trade with Europe was quickly killed by the ill-advised export embargo of last summer.

Export trade is desirable at the favourably situated Nova Scotia mines, and will probably develop later, irrespective of conditions in Europe that are now passing and are losing some of their urgency. The record of the operators of Nova Scotia is sufficient to show that they have always paid first attention to the home market. No Nova Scotian coal has been offered for sale in the Montreal market since about 1916, but it will appear again in the summer of 1921.

In Europe, the nationalization of coal mines is a burning political question. Millions of men, including not a few Canadians, have fought and died for the coalfields of Europe. The coalfield of the Maritime Provinces is a lone and singular deposit in all of Canada for a space of 1,800 miles. In this distance it provides the sole source of all the means of modern national defence that we have any right to call our own. It is producing, and has produced for six years, less than it did in 1913: yet the only definite action taken in regard to this incalculably precious national asset during this whole period has been the imposition of control and embargo.

The record is one that shows, not indifference, because there is no necessity to apply such an epithet to men who have doubtless done their best to pilot the ship of State, but it does show, plainly, that the primary national importance of coal supply has escaped the attention of our statesmen.

As to the "Gap of Ontario" there is little to be said except to urge the lessening of the gap by the maximum extension of the radius of distribution of the eastern and western coalfields, and to support any favourable engineering opinion on the feasibility of a deepening of the St. Lawrence waterway.

In regard to the Western Coalfield, in which is included the continuous deposits of Saskatchewan, Alberta, and British Columbia, and the detached fields of Vancouver Island and Northern British Columbia, the conclusion cannot be avoided that here lies the future workshop of Canada.

In his classic work on the "British Coal Trade," Prof. H. Stanley Jevons, writing, before the war, in regard to the untapped coalfields of the world, stated:

"We have to recognize that the Malthusian check to the increase of population operates in a totally different manner when the people of any region change by the aggregation of capital and spread of education from a purely agricultural community to a manufacturing nation with an organized modern commerce. A great stimulus to such a change will come wherever coalfields exist; and the population will thrive and develop upon the coalfields of the world, almost like flies upon honey."

Edwin C. Eckel, in an important recent book on "*Coal, Iron, and War*", points out that under modern conditions,

given the possession of raw materials, the industrial growth of a community need not be slow, but may even be compressed into a period so short as ten years.

The application of these two mature opinions to the conditions in the Canadian West emphasizes the obvious. Hitherto our dependence on extra-national sources of fuel has been associated very largely with the industrial eminence of Ontario, which is based on Pennsylvanian coal; but the new orientation which we see in the West promises an infinitely larger industrial growth over the western coal beds. Here, in the writer's opinion, will be founded the great pillars that will in the days to come support an industrial fabric transcending all present imaginings, and will, if wisely guided, prove "the most stable guarantee of our political permanence as an independent people." The coalfields of the West are large enough to build a nation on.

The most hopeful sign in this connection is that the people of the West have realized their coming industrial importance, and those who were privileged to attend the Winnipeg Meeting of the Institute realized that, whatever may be the case in the East, there has been formulated in the West that first requirement for a successful coal industry, what we will call a "coal conscience" among the general public.

It is evident, however, that rapid as may be the growth of population in the West, the outlet for coal here is smaller than the existing capacity of the collieries, unless an export trade is found possible. British Columbia is favourably situated, and its export trade is growing and is assured by the lack of good quality coal, except on Vancouver Island, along the whole length of the Pacific seaboard of North and South America.

The possibility of exporting Alberta coal to the States lying immediately south, and to outside points *via* Vancouver, is not by any means a remote one, and the extent of this trade is limited only by the market available.

The writer suggests that the present lop-sided arrangement of coal interchange between ourselves and the United States may be restored to a point less endangering to national safety, and less humiliating to national pride, and at least to the fifty-fifty basis that formerly obtained, by the following procedure, that is submitted as possible, namely,

(a) Complete independence of imported coal in the Maritime Provinces and Quebec, and in part of Ontario, achieved by enlargement of the Nova Scotia production.

(b) A moderate export trade from Nova Scotia to overseas points.

(c) Complete independence of imported coal in all Canada west of, say, Fort William, or Fort Francis, achieved by enlargement of transportation facilities, and recognition by the western railways that coal has become a permanent and growing feature of rail transportation in the West, both eastwards and westwards of the Alberta bituminous areas.

(d) Organized effort to secure export markets for Alberta coal, both southwards into the States, and through the ports of Vancouver and Prince Rupert.

(e) Enlargement of the export market of Vancouver Island coal.

## DISCUSSION.

MR. JAMES MCEVOY: It was time to bring up the subject of Canada's coal supply in a comprehensive way and we are obliged to Mr. Gray for his interesting paper. It is well worthy of careful consideration, and as it is written in such a manner, no doubt intentionally, as to invite discussion, the opportunity may be taken to offer a few remarks. The problem is a fitting one to bring before this Institute, as many of our members are large users of coal in mining and metallurgical plants all the year round, and others are intermittent users in auxiliary plants.

The coal supply of Ontario has to come from a distance which makes the freight rates a very large percentage of the total delivered cost, and it therefore behooves us to enquire as to what kind of coal we can afford to carry such a distance. A comparison of the relative effective heating values of different classes of coal may add something to the general data on the problem and will help to dispel an erroneous idea, prevalent in the minds of consumers who do not study efficiency, that a ton of coal is a ton of coal and that that is all there is to say about it.

Leaving the question of the distribution of our eastern coal by merely saying it is a good sound coal and ably dealt

with by Mr. Gray, let us pass on to the other possible source of supply in the west. The inferior brown lignites of Manitoba and Saskatchewan may be ignored as out of the question for use in Ontario; a small production of semi-anthracite and a possible future supply of anthracite may also be passed over.

This leaves us the bituminous and sub-bituminous coals of Alberta and of the interior of British Columbia. These coals can be divided into three classes, and while there is quite a range of variation in quality within each class, still they can all be assigned definitely to one or other of the three.

The three classes are as follows:

*Class No. 1.*—High-grade bituminous coal from the Kootenay formation.

*Class No. 2.*—Sub-bituminous coals from the Belly River formation.

*Class No. 3.*—Lower grade sub-bituminous coals from the Edmonton formation; these were formerly called lignites and are still so designated by most people.

The proximate analysis, including the moisture content, of these coals can be found in the publications of the Geological Survey and need not be dwelt upon here. There is a wide difference in the practical heating value of the three classes. A fair statement of the relative effective heat in the classes Nos. 1, 2, and 3 may be given as 9, 6, and 4½ respectively. There does not appear to be any reason to infer that this proportionate value is too hard upon the lower grades. It is based upon the number of pounds of water which one pound of coal will evaporate at a pressure of 150 pounds per square inch under ordinary working conditions in a locomotive boiler.

The figures represent the better grade of each of the classes, and it may be here definitely asserted that the figure 9 for Class No. 1 is not too high.

These figures being admitted, the conclusions which follow are inevitable. We can deduce from them, and from the freight rates, the relative distances we can afford to haul the three coals.

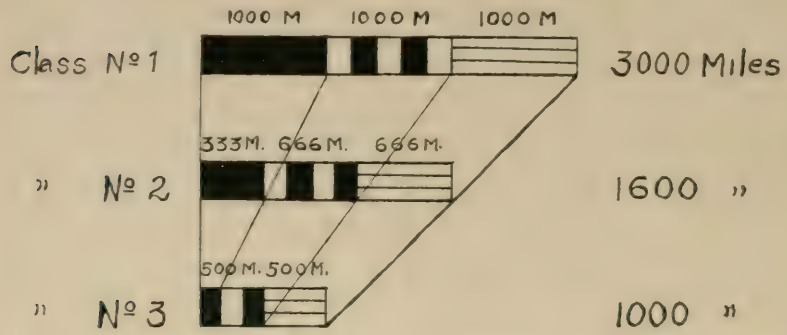
Assuming a price of $5.00 a ton at the pit mouth for all three coals, and a freight rate of one-half cent a ton-mile, we find that:

*First.*—We can afford to buy Class No. 1 and pay freight on a haul of 1,000 miles and still get the same value per dollar expended as in buying Class No. 3 without any freight charges.

*Second.*—We can afford to buy Class No. 2 and pay freight on a haul of 333 miles and still get the same value per dollar expended as in buying Class No. 3 without any freight charges.

*Third.*—If we haul Class No. 3, 1,000 miles we can afford to haul Class No. 2, 1,666 miles and we can afford to haul Class No. 1, 3,000 miles.

The following diagram will show this in greater detail:




Looking at the matter from another point of view we get results which are more startling. Assuming as before a price of $5.00 a ton F.O.B. mines for Class No. 1, then with equal haulage distances for all three classes, how much can we afford to pay for classes Nos. 2 and 3, at their respective pit mouths, in order to get the same value per dollar expended in freight charges and first cost?

It is obvious that as it will require a larger tonnage of Classes Nos. 2 and 3 to do the same work as Class No. 1, we cannot, therefore, afford to pay as much for the lower grades at the mines.

The answer is found by a simple calculation, and the following table shows the results:

| | Without any freight charges | After paying freight for 500 miles | After paying freight for 1,000 miles |
|---|---|---|---|
| Pit-mouth value of Class No. 1..... | $5.00 | $5.00 | $5.00 |
| " " " No. 2..... | 3.33 | 2.50 | 1.66 |
| " " " No. 3..... | 2.50 | 1.25 | 0.00 |

If anthracite coal were included in the comparison it would only mark more clearly the difference in values, for, although

it is not necessarily higher in B.T.U's, it gives more effective heat when burned in a suitable way, and there is less waste of unconsumed fuel up the smoke stack. At the other end of the scale, if the brown lignites were included they would greatly accentuate the already wide range of difference in value.

It must not be inferred from the foregoing that coals of Classes Nos. 2 and 3 do not serve a good purpose in the country. They are very well suited to domestic use where convenience of use is a large factor. They are generally cleaner to handle and with them a fire can be built up quickly. For these reasons many domestic users prefer to use these coals rather than a high-grade bituminous coal, especially if the latter cokes.

Having by the foregoing shown that we have in the west coals that are good travellers, the task of making a contribution worth while to Mr. Gray's paper would be incomplete without taking up the question of what is necessary to be done in order that Ontario consumers may avail themselves of this fuel supply. These coals are mined along the C.P.R. and the National lines of railway, and it is simply a matter of freight rates that determines how far they can supply the eastern demand. Now freight rates are supposed to be based upon cost, and the principal part of the cost is made up of locomotive cost and wages to engine and train crew.

It is not too much to hope that under certain conditions these train operating costs would be reduced to one-half for this particular traffic.

These conditions are:

(1). Powerful freight locomotives.

(2). Maximum train load consistent with efficient operation.

(3). Unbroken through trains.

If a locomotive and crew can handle 50% more tonnage, the reduction in cost is obvious, and if unbroken through trains are hauled there is no loss by shunting and making up trains at divisional points. The latter loss is an appreciable one, involving, as it frequently does, a lot of overtime at higher rate of wages.

If, under such a method, the freight charges could be reduced, the volume of traffic would increase and so make a further reduction in costs by a better distribution of fixed charges.

The consumer will have to play his part in the matter by placing his orders in such quantities and at such times as will make it possible for the railways to establish such through full-load traffic.

Our national railway, specialty over its G.T.P. line with its easy grades, is well adapted to specialize in such freight and could render a great national service by so doing. It may be mentioned that one railway in the western part of the United States did make a specialty of reducing the cost per ton-mile, and with marked success. By increasing the power of its locomotives and maintaining full loads it established, I believe, a continental record in that item. There is no evidence, in the case of this company, that the public profited by the reduction in cost, but in the case of our own national lines of railway the public must profit by any reduction of cost and it is to be hoped that the consumers would be specially benefitted to the extent of the greater part of any reduction effected.

MR. W. HERD: Mr. Gray uses the words "domestic coalfields" in a very general sense. The quality of the various domestic coals has not come in for consideration. No distinction is made between the high-moisture lignites of Saskatchewan, the sub-bituminous coals extending over a large territory west of Edmonton, and the true bituminous coals of Nova Scotia, Crowsnest district, and Vancouver Island.

The vast coal reserves of the west are lignites or sub-bituminous coals, which, in their raw state, are very unattractive to the habitual users of good bituminous coals, and at the present time it seems almost futile to force their use in the raw state. Economically, they cannot compete with bituminous coals, even if the latter have to be transported many hundred miles further.

As the known true bituminous coal areas of Canada are of limited extent, and as these areas are at present fairly well developed, this class of coal cannot be very materially increased — certainly not to the extent of replacing all imported coal.

Mr. Gray thinks the production of Nova Scotian coal might be increased to ten million tons annually. This no doubt is possible, but it is questionable if it is desirable. The Nova Scotian coal deposits are limited insofar as cheap production of the better quality coals is concerned, and an annual increase to the figure named would deplete the resources to an extent that cost of production would very materially increase in the not distant future. It would also deplete the metallurgical coal resources to an unjustifiable extent, and, due to the more extensive simultaneous working of superimposed seams, would result in considerable loss of coal in mining.

It would appear that the hope of making Canada nationally independent in her coal supply lies in the discovery of a colloidal fuel in which powdered lignite would be the basis, the cost of which, including mining and manufacture, will exceed by very little the cost to mine bituminous coals, and whose thermal value must compare favourably with that of a fair grade bituminous coal. The nation would be justified in sparing no expense to arrive at this result, as such a fuel would be suitable not only for home consumption but for export; although it is hard to imagine that the States to the south of these western lignite deposits would be importers, as they also, in many cases, contain vast resources of this class of coal.

The "Gap of Ontario" and parts of Quebec seem hard to bridge, due to their proximity to the United States coalfields, but probably a greater independence could be established in these provinces by the extended use of hydro-electric power.

Mr. Gray is to be congratulated in placing before the Institute such a clear statement of a condition which all interested in the welfare of Canada should strive to remedy.

MR. LOUIS SIMPSON: Mr. Chairman, might I venture to say a few words. I make no apology for taking the time of the members, because it is on record in the BULLETIN that our banks are already giving their attention to the more efficient utilization in Canada of Canada's natural resources, instead of letting the material leave the country in the raw state. Further, Mr. Gray, in a paper already referred to, has stated that Canada's coal supply problem is one that should be studied from the standpoint of national independence and defence.

To my mind the question, not only of coal but of Canada's fuel supply generally, is the most important thing before

Canada to-day, and I am surprised that this Institute, composed as it is of intelligent men, should not have taken this matter up sooner. That in our two leading provinces we are dependent for our fuel on foreign supplies is, to my mind, very bad business.

I have been endeavouring to arrive at the quantity of coal imported into Canada annually. Yesterday, you will remember, I called attention to certain discrepancies. I wanted to arrive at a figure as to the Canadian imports of coal that at least would be a safe basis to argue upon.

I, to-day, do not know whether the correct figure is 18 million tons or 20 million tons. Take the importation at 20 million tons. I am informed that coal laid down at the mills to-day costs $10 a ton. Twenty million tons, therefore, would mean 200 million dollars a year going out of the country. Think what it would mean if those 200 million dollars a year were earned and spent within Canada instead of being sent out of the country.

But that is not all. You must consider, along with the coal, the question of oil, and in this matter I must take very great exception to Mr. Gray's paper. It is not sufficient to consider Canadian requirements of coal as fuel unless Canada's requirements of oil are also noted.

The total oil importations into Canada for the last year would have a value of 50 to 60 million dollars, making a total of over 250 million dollars going out of this country year by year. What doesn't it mean to Canada, to the mining industry, to everyone here? Yet we are sitting here quite complacent, and are happily dreaming about copper and cobalt and nickel and goodness knows what.

To my mind the first thing we have to think about is our fuel supply — not coal alone, but our whole supply of fuel. I take further exception to Mr. Gray's article. He has also ignored the question of hydro-electric power. He has stated that the prosperity of a country can be measured by its coal consumption. I deny that. If you use hydro-electric power in place of coal, you may be doing better business. The coal, once burnt, is lost for ever; but hydro-electric development remains for all time.

We therefore have to look at this question in a broad way; not to look at coal alone, but to consider electric power, oil, and other possible sources of light, heat and power.

There is another feature of Mr. Gray's paper I did not like. Mr. Gray is a councillor of the Institute, and an engineer, and it is of course natural to expect that his paper would be considered and presented from the standpoint of an engineer. It seemed to me, however, that whilst "the hand was the hand of Esau, the voice was the voice of Jacob". It seemed to me he was writing as the special advocate of the coal operators, and therefore he ignored all the other sources of light, heat and power. Why did he not mention the possibilities of oil as a fuel, and of hydro-electric development as a source of power? Why come before you under false pretences? I feel this way about this question. Canada has been content to sleep in years past, because geologists told us Canada contains a large proportion of the coal of this continent. Canada forgot to remember that much of that coal is where it cannot be at present used. I think we ought, in this discussion, to divide the requirements of eastern Canada from those of western Canada. We may, perhaps, live to see the time when the industries of Canada will cease to be located in the provinces of Quebec and Ontario, and be located in the Rocky Mountain region.

I do think the situation is most serious. In Nova Scotia Canada certainly has a supply of coal, but a great deal of it is submarine and nobody knows certainly how much of that can be mined, or what the cost of mining will be. When you get under the sea you get in a pretty dangerous mining zone.

A MEMBER: Not at all.

MR. SIMPSON: Have you been there? How do you know what difficulties you will have to meet when you get a mile further out?

A MEMBER: We know what is done in other countries.

MR. SIMPSON: So do I, but I know from engineers that submarine mining has always been considered to be dangerous mining, and every quarter of a mile you go from the pithead, the mining will cost you more. You can't sink a shaft in the middle of the sea. The shaft must be on the mainland. I have that information on the *ipse dixit* of an engineer who has spent most of his life mining coal.

There is another question which is also of importance,

and I think we should consider it. I am quoting from a writing which came into my hands, and I understand it has been sent round to different members of parliament and senators. It was given to me by a senator. Speaking of the advantages of Sydney, it is stated that Sydney's advantages are shown by the presence of large beds of coal, and by the existence of large iron works and chemical plants; and the article proceeds: "No modern nation can obtain economic independence, unless it possesses within its frontiers bituminous coal. Coal is the source of the chemical elements that are the basis of the most destructive military ammunitions. It has been said by a German military writer that victory will go to the nation that can mine and carbonize most bituminous coal." Therefore, if this statement be correct, it seems to me it is of great importance that the Nova Scotia coal deposits should not be depleted, nor do I see the wisdom of increasing the production of coal pits in Nova Scotia. In case of war, which I hope we will not see, but which may come nevertheless, our source of the necessary explosives might be in the far west.

I desire to call attention to another matter which is of interest to every member here. In the same paper there is this statement: "Iron, limestone, coal, and coke can be produced as cheaply as anywhere in America. Steel production at Sydney has a strong initial advantage against the rest of America" Every engineer has to buy and use iron and steel. If the statement made in this paper is true, you ought to be able to buy mining machinery made of Sydney limestone, iron, and coke as cheaply as you can buy similar machinery in the States. If it be true that iron, limestone, and coke are produced as cheaply in Sydney as anywhere on this continent, there is then no reason why you should pay more for iron and steel in Sydney than you would have to pay at Pittsburg.

What are the facts? If you want to buy a machine, you are compelled to buy it in Canada or in the United States. If you buy it in Canada, the machine will cost you the price asked in the States plus duty, freight, and cost of exchange.

You have to dispose of your products in the markets of the whole world, yet you are handicapped by having to pay 40 or 50% more for machinery than your competitors pay for it.

Now I think the members of this Institute ought to put themselves on record that any machine required for their use, and not made in Canada, should be allowed to be imported into Canada without payment of duty. They have these advantages in certain of Canada's industries. The textile people only pay 10% duty. The beetroot sugar makers pay nothing. And there are quite a number of industries that are allowed to import machinery without the payment of duty.

I would ask, why should the mining industry be thus penalized? Yet it is even worse than I have stated.

THE CHAIRMAN: Excuse me, Mr. Simpson, for interrupting you. Our time is limited and I must invite speakers to devote themselves to the subject in hand. The question of duty on mining machinery is very important, but we are not discussing it this afternoon.

MR. SIMPSON: I am finishing right away. But if some of our industries get machinery in free of duty, why should not all of us?

MR. GRAY: Mr. Simpson's point that my paper ignores fuels and sources of power other than coal is not well taken and is irrelevant to the scope of the paper. It is the third of a series read before the Institute, all of which confined themselves to a discussion of our national *coal* problem. In the first paper, read at Toronto in March, 1920, I stated:

> "The writer will not attempt to discuss what relief we can obtain from utilization of our peat-bogs and hydro-electric reserves, for these have been ably dealt with by specialists, and even their completest utilization can only be in the nature of a palliation of the fuel problem, and can never constitute a remedy".

As to the possibility of depriving Canada too rapidly of its natural resources, if Mr. Simpson has read my paper, he would have noticed that I pointed out the limited extent of the coal reserve of Nova Scotia and stated that an export trade "is probably not one to be encouraged to a point of national deprivation, but it has never reached that point".

# THE DEVELOPMENT OF THE COAL INDUSTRY IN CANADA.[1]

By F. W. Gray.

Annual Western Meeting, Vancouver, November, 1922.

In previous communications to the Institute, the writer has developed at some length the necessity the Canadian people are under to preface all the ordinary technical and economic considerations relating to coal mining by primary and more urgent considerations of a national and political nature.

A re-statement of this necessity would make this paper of undue length, and, for the writer's views, reference may be made to the original papers[2].

It is necessary to state, however, that the argument which follows is based on the assumption that Canada should be supplied with coal from Canadian mines, as far as possible.

As a concrete example, and to focus this viewpoint, quotation may be made from an official statement of the Government, given out from Ottawa through the Canadian Press on 29th May, 1922, namely:

"The government stated that preference was given to Canadian coal, "but that where American coal, including duty, could be delivered at certain "Canadian points *more cheaply than Canadian coal* the railway management "took the attitude that they would continue to purchase to the best "advantage."

This official expression is also a fair statement of the attitude of the coal-purchasing public in Canada. It is an

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[1] Prize Paper, President's Competition, 1922.

[2] "Canada's Coal Supply," Transactions, 1920, pp. 297-307; Annual General Meeting, Toronto, 1920.

"The Part of the Western Coalfield in Canadian Development," Annual Western Meeting, Winnipeg, November, 1920.

"Canada's Coal Problem." This volume, pp. 293-309; Annual General Meeting, Montreal, 1921.

attitude to which no exception can be taken if the questions associated with maintenance of national independence are ignored; and it is probably a correct attitude if the district within which American coal is used is outside the economical radius of distribution of Canadian-mined coal.

The principle is, however, a pernicious and disintegrating one if applied to any portion of Canada where the use of American coal displaces Canadian coal and causes a restriction of the expansion of Canadian coal production. Unfortunately for the development of coal mining in Canada, the principle has been so applied, and large Canadian purchasers — the Government railways included—have so used their ability to purchase American coal, as to rigidly restrict Canadian coal within the hitherto customary and assumed limits of its distribution, and there has not been any convincing display of energy to extend the radius of distribution of Canadian-mined coal to the utmost possible.

The large industrial purchasers, the Government, and the people of Canada, have bought coal where it was cheapest in money value, not realising that such a policy has been excessively dear and nationally subversive, inasmuch as it has injured Canadian miners, has restricted employment in Canada, and has made our country almost ludicrously dependent upon the United States for what is undoubtedly the primary and most essential raw material of civilization, defence, and national persistence.

The conclusions put forward in the following argument will, of course, not be sustained unless it is first admitted that coal brought into Canada is dear at any price if it displaces Canadian labour, and, by rendering our country unnecessarily dependent upon outside sources for coal supply, endangers our political autonomy.

## ANTHRACITE SUPPLY.

Canada shares with the United States the unique luxury among the nations of anthracite as a domestic fuel. But not having true anthracite in the Dominion, or coals of anthracitic nature situated near large centres of population, it is necessary for us to import anthracite. The disadvantages of such

dependence are sufficiently in the public mind at this time not to need emphasis.

Apart, also, from the temporary stoppages of anthracite supply that may arise from time to time, is the inescapable fact that the anthracite reserves are so small that definite signs of exhaustion have already appeared. Furthermore, a steady increase in the cost of anthracite year by year is inevitable.

Anthracite is admittedly a luxury, and will, within a period that is so short as to concern this generation of Canadians most vitally, be virtually beyond the reach of all but the very rich. The substitution of bituminous coal for anthracite in Canada will, from now on, rapidly increase. The ingrained habits of a population accustomed to a cleanly fuel, the construction of the furnaces used in domestic heating, the restricted flue areas, and a natural repugnance to smoke and dirt in our cities, suggests that bituminous coal will be treated to obtain a smokeless fuel with long heat-duration and minimum ash-content in direct substitution for anthracite.

The production of such a fuel from bituminous coal has for long been the goal of many experimenters, but as yet no thoroughly satisfactory fuel has been placed on the market in large quantities.

The most practicable substitute, at this date, is probably a dense coke. This could be made from bituminous slack coal in by-product coke ovens. By the use of the modern by-product oven, which is designed with narrow oven-chambers and operates on, say, a sixteen-hour coking period, and utilizing moderately priced slack coal, it is possible to produce, at a moderate cost, a coke that is an acceptable substitute for anthracite. The credits from by-products recovered and the surplus gas materially cheapen the cost at which the coke can be sold. The surplus gas is available for sale for heating and lighting purposes, and where the coke plant is situated near a large city could be sold to gas companies for distribution through existing mains. The practice is already a well established one.

In natural-gas districts, such as are found in some parts of Ontario, where the gas supply is failing, this could be fortified by coke-oven gas, or, in the event of entire failure

of the natural supply, the existing system for distribution and consumption could be utilized.

The erection of coke-oven plants near large cities is, as stated, a probable development of the future. In those parts of Canada within the radius of distribution of bituminous coal mined in Canada, such as Montreal, it should be possible to gradually transfer to Canadian producers of bituminous coal all the market now represented by the use of anthracite, if full use is made of the opportunity presented by the coming demand for an anthracite substitute.

Anthracite is mentioned here, before bituminous coal questions are touched, to show its relative unimportance to Canada. In any long-range forecast of coal mining development in Canada, anthracite may be passed over as not being a permanent factor.

## BITUMINOUS SUPPLY AND DEMAND.

Question of supply and demand of bituminous coal in the Dominion fall into three natural divisions, namely:

(a) Area of economic distribution of Maritime Province coals.
(b) Area of economic distribution of Western coals.
(c) Intervening area, containing no coal seams.

The extent of the coal-less "Gap of Ontario" is the real national problem of coal supply. To what extent is it desirable to widen the radius of distribution of eastern and western coals, and thereby narrow this gap? To what extent is the populous centre of Canada justified in taking advantage of its geographical projection into the territory of the United States and the ganglion of water and rail transportation that centres near the Great Lakes? There is a mean which can be worked out, by which this country can save itself from too entire dependence on the United States for coal supply, and at the same time not lose the advantage certain parts of Central Canada possess in their proximity to the area of industrial activity that is energized from the great central coal-field of the United States.

What is this mean? The following distribution of bituminous coal is suggested as one that might be attempted.

*Nova Scotia Coal:* Should supply all the coal requirements (including anthracite substitutes) of the Maritime Provinces, Quebec, and that portion of Ontario comprised in the triangle lying between the St. Lawrence and the Ottawa rivers, east of a line drawn from Brockville to Ottawa. Should also supply the district served by the Transcontinental Railway, at least as far west as Cochrane, with coal sent to Quebec by water and thence trans-shipped for western forwarding.

*Western Coal:* Should supply all Canada west of a point between Fort William and Winnipeg, say Fort Francis.

## NOVA SCOTIA COAL.

As to the relation between available supply and demand, it is not probable that the coal deposits of Nova Scotia can be usefully or advisedly developed to produce more than ten million tons annually. That is to say, ten million tons annually is the maximum output which can be maintained over a sufficiently long period to allow amortization of the investment necessary to develop the coal deposits to this capacity. In other words, the coal deposits of Nova Scotia, estimated, with present knowledge, to contain between 7½ and 10 billion tons of coal, are not unlimited.

There is a possibility, amounting almost to a probability, that some concealed coal-fields exist in Nova Scotia, but any coal seams which are yet unproven must be at such depths as will require heavy expenditures to develop. Such possible reserves, for example, as may exist to the south-westward of the Springhill field, or as may be gained by submarine working of coal seams not now regarded as accessible, would serve to lengthen the period of maximum yield of the Nova Scotia reserves as a whole, but would not increase the practical maximum annual output herein suggested.

The market for Nova Scotia coal has not, in recent years, exceeded 5¼ million tons annually, for which low figure both a lack of capacity to produce and a lack of ability to make sales are jointly responsible.

Both conditions are the result of the war, loss of production capacity being chiefly due to a lessening of the available number of skilled miners through enlistments.

Inability to make sales has affected Nova Scotia in the same way that it has affected other coalfields as a consequence of generally lessened consumption of coal in industry, but the condition has been accentuated in Nova Scotia by temporary loss of the St. Lawrence market, by high production costs associated with high wages, and by the acute depression in the steel industry, all natural consequences of the war. Present conditions must, therefore, be regarded as unrepresentative of those which may be looked for when the world settles down, and when Canada resumes her interrupted development by permitting immigration and fostering domestic industry.

A comparatively new and powerful competitor along the Atlantic seaboard is coal from West Virginia. The large amount of cheaply mineable coal in West Virginia, and the low price at which coal of first-rate quality can be laid down at Norfolk, has introduced a new feature into the North American coal trade which is upsetting many long-established precedents.

The following is believed to be a fair appraisal of the available market for Nova Scotia coal — presuming normal operation of steel plants and the supplying of 75% of the coal requirements of Quebec province.

| | *Long Tons per annum* |
|---|---|
| Railways | 1,500,000 |
| Steel Plants | 1,500,000 |
| Province of Quebec | 3,000,000 |
| Maritime Provinces and Newfoundland | 1,500,000 |
| Export and Bunkers | 500,000 |
| Used at Collieries | 800,000 |
| Total | 8,800,000 |

The quantity allowed in the foregoing table for bunkers and exported coal is not large, and has often been exceeded. A larger figure is not used because it is desirable that the limited supply of coal in Nova Scotia should be looked upon as primarily for Canadian use.

The production of coal in Nova Scotia reached its hitherto recorded maximum in 1913 when the output was 7,263,000 tons, but for the past six years it has varied between 5,000,000 and 5,800,000 tons. It is unlikely to exceed the 1913 figure before the expiration of five to ten years from this date, so

that it is fair to consider coal mining in Nova Scotia as having attained almost full development and as unlikely to experience any spectacular enlargement in the future.

The industry is an old one, probably the eldest mining enterprise of the Dominion, and it has passed through all the infantile troubles that new fields must experience. By consolidation of the coal companies there has been laid the foundation for the financial stability and for the unified and continuing management that in the past the field has so conspicuously lacked.

Dealing with coal mining in Nova Scotia, therefore, in the terms of "progressive mining" which the President of the Institute has prescribed, it may be described as having entered a second stage. The first stage was distinguished by virgin land areas containing thick and accessible seams, operated with divergent purposes by competing companies having meagre equipment and little financial stability, limitations which were offset by the cheapness with which coal could be mined. The second stage is marked by partial exhaustion of the thick seams, and preponderance of extraction from submarine areas, limitations offset by unified purpose, modern equipment, and much greater financial resources, but accompanied by higher production costs.

The progress of this second stage has so far been accompanied by more complete use of inferior coals, by better preparation of coal for market through screening and washing, by extended use of electricity and coal-cutting machinery, and by the speeding up of underground transportation. A much greater use of electricity below ground is probable, and its freer application to removal of coal from the face to the surface will be required if Nova Scotia coal is to compete with coal mined in the United States, where the mechanical side of coal mining has been more highly developed than elsewhere.

It is probable the great submarine coal-field off the Sydneys has a seaward extension far exceeding the distances at which coal has hitherto been won from under the ocean. New methods of transportation and new applications of electricity are likely to be devised to meet this unique condition, so that it is fair to assume the progress of the practice

of undersea coal mining in Nova Scotia will, for many generations to come, present interesting developments.[1]

Nova Scotia coals are all a fairly consistent grade of bituminous coal, suitable for all domestic and industrial uses with but little need for discrimination between them. For metallurgical purposes the quantity of low-sulphur coal is not in excess of requirements, but this is by no means a condition peculiar to Nova Scotia.

The complexities experienced in the West concerning the grading of coals, varying from lignite to high-grade steam coals, and the influence of this condition upon marketing and transportation, are not present in Nova Scotia. The even quality of the coal and the concentration of management tend to general uniformity of the grade of coal offered for sale.

## WESTERN COAL.

The first characteristic of the coal industry in the West, viewed in terms of progress, is its extreme juvenility and the small annual production obtained in relation to the number of mine openings, and the enormous reserve of coal available.

It is necessary, however, to deal separately with the coal industry on Vancouver Island, where the conditions, in many respects, resemble those of Nova Scotia, and where the industry is of long standing.

The output of Vancouver Island has varied in recent years between one million and 1¾ million tons annually, a production that is much smaller in relation to the calculated reserves than that of Nova Scotia. The coal seams of Vancouver Island do not possess the same regular contour and persistency as do those of Nova Scotia, and it is probably fair to state that on Vancouver Island, as in Nova Scotia, the industry has entered upon a second stage and that no spectacular increase in the annual output is to be looked for. It is conceivable that production might be worked up to 5 million tons annually, but the coal deposits, as was noted in regard to Nova Scotia, are too limited to warrant the investment of larger sums of money than are required to create and maintain an annual output exceeding 5 million tons.

[1] See "Coal Fields and Coal Industry of Eastern Canada", Mines Branch Bulletin No. 14, 1917, pp. 31-35.

Vancouver Island coal has, in addition to the bunker and export business natural to its maritime location, supplied certain requirements of the mainland of British Columbia up to the point where it met the competition of coal coming from the Rocky Mountain fields and Alberta, but the ability of these last named fields to extend their markets to the coast will grow from year to year, and it seems probable that Vancouver Island coal will find its most permanent and profitable markets in bunkering, and in exporting coal to the cities of the Pacific coast of North and South America.

The conditions under which coal is mined on Vancouver Island make for a high cost of production, which limits the ability of the coal companies to compete with coal coming from Utah and other Western States or with the low-grade lignites of Washington; and in this respect the problem of the Vancouver Island operator is not unlike that of the Nova Scotia operators. Heavy initial capital expenditures are required to enable coal to be cheaply mined on Vancouver Island, particularly in those districts where the coal seams are disturbed and dirty. Consolidation of coal-mining companies has been partially effected on Vancouver Island, but the coming years will probably force more complete consolidation of financial interests and concentration of managerial energy upon those areas where coal can be most profitably mined for sale.

Neglecting, for the purposes of a review of this general character, the detached occurrences of coal scattered over the enormous area of the province of British Columbia, the most important deposit of coal in Canada is contained in that vast extent of coal-bearing rocks which stretch from Estevan, Sask., to the crest of the Rocky Mountains, and which lie along the eastern flank of the latter for six hundred miles in Canadian territory.

In this wide extent of country, covering every extreme of topographical outline, to a large extent uninhabited, and only partially prospected, is found a variety of fuels ranging from low-grade lignites to coals that approach anthracite in their carbon content.

This varied coal-field has been attacked at points where settlement or the presence of railways has caused a local

demand for coal. The investment of capital has been, so far, of a very independent and detached character, and it is not too much to say that coal mining in Western Canada has, up to recent date, been marked by multiplicity of points of operation, causing unwise competition and general frittering away of both investment and human energy. In the same manner a multiplication of grades and sizes of coal offered for sale has taken place, a procedure that always entails waste of run-of-mine product and a general lowering of the average selling price of the coal, to say nothing of mystification of purchasers and the complexities introduced in all records and business transactions.

Lately there has been a tendency to co-ordination of purpose among the coal operators, brought about by such organizations as the Western Canada Coal Operators' Association, which will, in due course, lead to a combined simplification and enlargement of vision as regards the essential oneness of the western coal-field, and to unity of aim in regard to the markets this vast coal reserve must obtain if it is to fill the place in Canadian life and national development that its supreme economic importance suggests it should fill.

There is no necessity, in considering the future of this western coal-field, to limit the maximum annual output and the amount of capital invested because of limitations in coal reserves. In Western Canada the limit is one of available markets, and the chief care of the investor must be to see that he does not provide a greater coal production than the market requires.

The present stage of the western coal trade is the result of agricultural settlement in localities decided not so much by possession of agricultural suitability as by proximity to transcontinental railways.

The second stage will come when, instead of the coal trade being a result of sparse and irregular agricultural settlement and railway coal consumption, the coal-field, itself, emerges as the *cause* and the source of manufacturing activities and as a magnet to draw population.

At this stage will commence the process of consolidating coal areas and corporate interests, leading to the formation of large coal companies holding title to mineable coal-areas

sufficiently large to secure interest payments on the ample investments that will be required to produce coal at moderate cost over long periods.

There is a tendency to take too short a view of coal-mining management and investment and to criticize corporations that hold large areas of coal "out of operation." Large companies, able to map out mining programmes and capital expenditures over periods exceeding an average lifetime, are necessary to economical operation and to stability of investment, employment, and selling prices. Large reserves of coal are essential to the success of such planning for the future.

Looking at the other side, a multiplication of mine openings, with small reserves tributary to each enterprise, inevitably leads to instability of all economic factors and to spoiling of a coal-field from the viewpoint of the mining engineer.

In all new coal-fields there is a tendency to first attack the coal seams that are most accessible, of best quality, and most cheaply mineable, and at any given time the coal that is of good quality and sold at moderate cost will find the readiest sale and will be transported for the longest distances.

In the western coal-field, the coals are of highest grade near the mountains and of poorest grade as they approach the eastern edge of the field. Except, therefore, as determined by local concentrations of population, which are the result of locations lending themselves to transportation needs in the preliminary agricultural stage of settlement — such as Winnipeg — the first large-scale mining of coal may be expected to develop where the metamorphosed coals in the mountains are crossed by railways and thereby made possible to develop, as in the Crowsnest district, Canmore, and Brulé Lake. These coals, ranging 60% carbon and over, will stand long-distance transportation, and in the first stage of development will successfully compete for any uses against coals of lower carbon content lying nearer the point of consumption.

Second in point of time to be thoroughly developed will be the sub-bituminous coals in the foothills, contained roughly in the area lying between the mountains and a line drawn from Lethbridge to Edmonton. In this area the lower carbon content of the coal is offset, when compared with the higher grade material found in the mountains, by greater accessibility

and ease of mining, and local industries of much magnitude are certain to develop in this area, based upon the use of locally mined coal.

East of Lethbridge, the carbon content continues to diminish, until low-grade lignites only are met with, and beyond this there is a complete absence of any coal resources. The territory last named will probably continue to import coal, irrespective of the local lignites, for all manufacturing operations, and preference will be given to the "mountain" coals over "foothill" coals because of greater fuel-value off-setting higher costs of rail haulage.

It seems probable, therefore, that "mountain" coals will come to be considered as export coals, and that "foothills" coal will form the basis of industry on the ground.

The beneficiation of low-grade lignites in the Saskatchewan field will, if proved to be economically successful, afford a substitute for anthracite in domestic heating. For this purpose it will compete with a fuel that will never cost less than it does to-day (money inflation allowed for). For steam uses, it will be difficult for lignites of low grade, either briquetted or powdered, to compete with the high-grade bituminous "mountain" coals, even with the freight handicap on these coals.

The production of the Western coal-field attained its highest level in 1920, as follows:

| | *Saskatchewan* | *Alberta* | *Eastern B.C.* |
|---|---|---|---|
| Lignite | 343,475 | — | — |
| Bituminous | — | 3,410,872 | 943,908 |
| Sub-bituminous and Lignites | — | 3,266,949 | — |
| | 343,475 | 6,833,500 | 943,908 |

Total, 8,120,883 short tons.

The disposal of the coal mined was as follows:

| | *Saskatchewan* | *Alberta* | *Eastern B.C.* | *Total* |
|---|---|---|---|---|
| Consumed in Canada | 318,987 | 6,180,585 | 228,253 | 6,727,825 |
| Exported to United States | — | 132,918 | 525,607 | 668,525 |
| Coke-making and colliery use | 24,488 | 519,997 | 180,048 | 724,533 |
| | 343,475 | 6,833,500 | 943,908 | 8,120,883 |

In the same year, also, the markets of the Western coal-field saw their widest extension, as succinctly set forth by the Director of the Mines Branch in a report for 1920.[1] In particular reference to Alberta, but in terms applicable to the Western coal-field as a whole, Mr. McLeish there comments:

> "The falling-off of over a million tons in 1919 was ascribed by the Chief Inspector of Mines in part to the accumulation at the end of 1918 of unusually large stocks in the hands of the trade or consumers. It will be noted also that less coal was shipped to points outside the province during 1919 than during the previous year. That portion of Central Canada comprising Saskatchewan, Manitoba and Ontario, west of the head of the Lakes, had been supplied annually with from 2,000,000 to 2,500,000 tons of coal imported from the United States. Restrictions on this supply during 1918 necessitated a larger consumption within the territory of Alberta coal. An effort was made on the part of the Alberta producers to secure and retain an increasing proportion of this market. In view of this, it is not surprising that the 1920 production reflected this effort by attaining its maximum record."

What the Western coal-field did in 1920, as a result oi efforts put forth under stress of war conditions in 1918, if can repeat and exceed in proportion to the growth of population.

The per capita consumption of coal in Canada is about 3.75 short tons, but west of Fort William it appears to be greater than the average for the whole country, being about 4.1 short tons. This is not an abnormal figure, and may be expected to increase. The smaller per capita consumption of coal in the East may be explained by greater availability and development of hydro-electric power, larger use of wood, and greater percentage of anthracite consumption. The climatic conditions of the Middle West, the lack of firewood in the prairies, and the general use of bituminous and sub-bituminous coals, account for the higher per capita rate of the West.

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[1] See "Production of Coal and Coke in 1920", Mines Branch Bulletin No. 567. p. 21.

Prof. H. Stanley Jevons,[1] in his monograph on "British Coal Trade", estimates a maximum coal consumption of 4.8 long tons (5.37 short tons) per head of population in Britain. The following table, taken from Prof. Jevons' work, is suggestive:

*Average Home Consumption of Coal in Britain.*

| Decade | Consumption (Millions of Tons) | Consumption Per Head | |
|---|---|---|---|
| | | Long Tons | Short Tons |
| 1855–60 | 62,385 | 2.775 | 3.1 |
| 1861–70 | 88,095 | 3.579 | 4.0 |
| 1871–80 | 112,386 | 4.047 | 4.6 |
| 1881–90 | 132,793 | 4.260 | 4.7 |
| 1891–1900 | 149,657 | 4.303 | 4.8 |
| 1901–1910 | 172,108 | 4.423 | 4.9 |
| 1911– | 184,852 | 4.527 | 5.1 |

The per capita coal consumption in the United States exceeds five tons.

The market for coal in the West depends more upon an increase in population than upon any other factor. The export possibilities include the supplying by rail of good-grade bituminous coal from the "mountain" mines to Montana, Idaho, Washington, and Oregon, and the coast-wise trade now held by the Vancouver Island collieries. There is good reason to expect that coal from the "mountain" mines may be shipped through the Port of Vancouver, either for ships' bunkers or as cargo. This development will depend on how successful the Vancouver Island collieries are in keeping down the cost of production.

The exports of Western coal have been, in recent years, as follows:

| | British Columbia | | | Alberta | Total |
|---|---|---|---|---|---|
| | Vancouver Island | Nicola and Princeton | Crowsnest | | |
| 1919 | 427,103 | 7,068 | 418,533 | 121,212 | 973,916 |
| 1920 | 339,703 | 1,639 | 535,607 | 153,610 | 1,030,559 |
| 1921 | 327,786 | 30,194 | 562,340 | 243,199 | 1,163,519 |

[1] "British Coal Trade," by H. Stanley Jevons; Kegan Paul, French & Trubner & Co., London, 1915, pp. 743-752.

## THE FUTURE OF THE COAL INDUSTRY.

The following table attempts to forecast the home consumption of the provinces of British Columbia, Alberta, Saskatchewan, and Manitoba, over the ensuing three decades. The under-mentioned factors are regarded as probable, viz:

(a) Steady increase in the per capita consumption of coal.
(b) Increase of population as shown.
(c) Limitation of imports of U.S. coal into the territory named, brought about by increased ability of Canadian coal to meet coal from the United States, which will steadily increase in cost delivered in Canada.
(d) Increase of rail exports of "mountain" coal to border states, and maintenance of Pacific markets supplied by Vancouver Island coal and an increasing percentage of coal from Eastern British Columbia.

*Forecast of Growth of the West Canadian Coal Industry based on assumed growth of population.*

| Decade Commencing | Population | Coal Consumption | | Coal Production | Export Coal | Imported Coal |
|---|---|---|---|---|---|---|
| | | Per Capita, Tons | Total Tonnage | | | |
| 1920 | 2,390,000 | 4.1 | 10,000,000* | 10,035,852 | 1,039,124 | 1,000,000* |
| 1930 | 4,300,000 | 5.0 | 21,500 000 | 22,000,000 | 1 500,000 | 800.000 |
| 1940 | 6,430,000 | 5.2 | 33,500 000 | 34,000,000 | 2,000,000 | 600,000 |
| 1950 | 8,000,000 | 5.3 | 42,500,000 | 44,000,000 | 2,500,000 | 500,000 |

*Available statistics show coal imported into the "Head of the Lakes", including Manitoba, and an accurate dissection of imported coal used in Manitoba and Saskatchewan cannot be made.

Thirty years ago the factors influencing the production of coal in the West were as follows:

Population..... 250,000 persons
Coal Output.....1,304,000 tons
Coal Exported..... 900,000 " (From Vancouver Island)

A little study of the figures will reveal the conservative nature of the forecast of population, and, if this is admitted, the remainder of the calculation naturally follows.

Another table, which follows, attempts to forecast the coal statistics of the Dominion, on the basis of expected population, and assuming, as in the first table, a steady increase in per capita coal consumption. Very little of the

increase in Canadian coal production forecasted is estimated as likely to come from the mines of Nova Scotia or Vancouver Island. The calculation is based chiefly on the expected population growth of the West, accompanied by development of the western mines to fully take care of the increased domestic consumption without increase of importations from the United States into the territory west of Fort Francis.

The increase of coal consumption in the East, consequent upon increase of population, can be partially met by increase of output from the Nova Scotia mines, but the calculation indicates that the quantity of coal necessary to be imported from the United States into the central area of Canada will not decrease, but rather increase. If the export of Canadian coal is increased beyond the modest limit of two million tons per annum, indicated in the table, the importation of United States coal will require to be correspondingly increased.

*Forecast of Canadian Coal Production, Importation, Consumption, and Exports, based on assumed Increase of Population.*

| Year | Population | Consumption per head, tons. | Total Consumption | Mined | Imported | Exported |
|---|---|---|---|---|---|---|
| 1886 | 4,324,810 | 0.75 | 3,480,111 | 2,116,653 | 1,884,161 | 520,703 |
| 1891 | 4,833,239 | 1.15 | 5,586,712 | 3,577,749 | 2,980,222 | 971,259 |
| 1901 | 5,371,315 | 1.81 | 9,722,877 | 6,486,325 | 4,810,213 | 1,573,661 |
| 1911 | 7,206,643 | 3.38 | 24,247,698 | 11,323,388 | 14,424,949 | 1,500,639 |
| 1920 | 8,700,000 | 3.76 | 32,731,521 | 16,631,954 | 18,657,741 | 2,558,174 |
| 1931 | 11,000,000 | 4.25 | 46,700,000 | 30,000,000 | 18,700,000 | 2,000,000 |
| 1941 | 13,500,000 | 4.75 | 64,000,000 | 42,000,000 | 24,000,000 | 2,000,000 |
| 1951 | 15,000,000 | 5.00 | 75,000,000 | 52,000,000 | 25,000,000 | 2,000,000 |

(All quantities in short tons.)

It is evident, if the coal trade of Canada develops on lines only approximating to the foregoing forecasts, that the whole fiscal outlook and the political complexion of the Dominion will have been radically altered within thirty years. Of a total of 15 million people, 8 million will live west of Winnipeg and there will be an important concentration of population in Alberta that will not be agrarian in outlook, but distinctly industrial, with all that this implies in a democratic country.

Of 52 million tons of coal estimated as likely to be mined, some 38 millions, or 73%, will be mined in Alberta, Saskatchewan, and Eastern British Columbia.

The West will become financially strong and politically powerful because of a combination of the two basal industries, agriculture and mining, that are not similarly juxtaposed elsewhere in Canada.

The importance of the coal-field of Nova Scotia will become more and more evident as the dependence of central Canada on an outside source for coal supply is evidenced by recurrent crises in coal supply similar to the one now being experienced; and the necessity to narrow the dependency of the coal-less "Gap of Ontario", by sending Western coal east and Eastern coal west, will emerge more and more clearly.

Up to the present, the scientists and geologists of Canada have been engaged, and will yet be engaged for many years, in recording and reporting upon the obvious outcropping evidences of coal deposits. The search for hidden coal-fields, which pre-supposes knowledge of conditions prior to the laying down of the coal-seams and the consequent earth movements, has not as yet been necessary in Canada, and as a consequence our knowledge of the coal reserves is incomplete. For the same reason, coal mining has not yet been prosecuted to depths greater than 2,500 ft., and that only in a few localities. The coal-seams so far mined have been selected because of their attractive quality, and we are not yet expert in the preparation of inferior coals for market. Proximity to the coal-fields of the United States, unrivalled in extent and quality, and familiarity with the lavish use of coal which this opulence has occasioned, has caused Canadian miners to despise coal-seams that some day will be operated with profit. Compared with

the United States, our coal resources lack some attractive features, but that is all the more reason for thrifty management and for using to best advantage the resources we have.

No attempt to outline the progress of coal-mining in Canada would be complete that did not make mention of the work of the Department of Mines.

The enquiries of the investor in coal-mining enterprises, and of the student of mining economics, have been lightened by the scientific reports and the statistics of the Geological Survey and the Mines Branch and, more recently, the Bureau of Statistics. The extent to which the development of coal-mining in Canada is indebted to these departments of the public service is very large, and there have been few expenditures of public funds in Canada that have yielded, and will yield, such valuable returns.

The general public does not realize the importance of the work done by Canadian scientists and mining engineers in preparation for the International Geological Congress held at Toronto in 1913, which brought Canadian coal resources before the whole world, and, by completing a survey of the world's coal reserves on a scale never before attempted, has made it possible to assess the relative importance of our own coal-fields and to find them somewhat reassuring. In the progress of coal mining in Canada, the monograph on the "Coal Resources of the World," issued at Toronto, marked a notable advance.

Generally speaking, it does not seem probable that Canada will become a dominating factor in the export coal trade of the world, although the country can play a respectable role in this way for an indefinite period ahead. Canada's chief hope of benefiting from coal deposits within her borders is by a combination of domestic industries and agriculture, and the inculcation of a wholesome fear among our people of too great a dependency on the United States coal-fields.

## FUELS AND SOURCES OF POWER OTHER THAN COAL.

It is necessary, in order to avert misinterpretation of the purpose of this paper, to explain that it is limited to the discussion of the progress and development of coal-mining in Canada, and that it does not treat of the related but distinctly

differing questions of the fuel and power supply of Canada, or the matter of substitutes for coal.

Broadly speaking, no conflict of interests is probable between the coal producer and those who seek to develop the water-powers of this country for the generation of electric power.

The fuel insufficiency of New Brunswick, Quebec, Ontario, and Manitoba is so pronounced that the completest and most efficient utilization of water-powers cannot begin to relieve the situation. Where hydro-electric power is available at advantageous cost, the coal producer himself will be first to avail himself of it in order to save the use of coal for power-raising at the mines, as is instanced by the hydro-electric development of the Canadian Collieries (Dunsmuir), Limited, at Comox, Vancouver Island.

Generally, the districts of Canada that contain the most abundant and dependable water-powers are precisely the districts that are devoid of coal-seams. The fullest development of both the coal seams and the water-powers of Canada cannot relieve us entirely of the necessity to import coal, and both can be encouraged without stint with no fear of either interest suffering, and without the slightest likelihood of meeting the power requirements of the country.

Similarly, the use of peat as a fuel, and the utilization of oil-shales, can be encouraged to the limit of their availability without any danger of affecting the production of coal in Canada, and also with but little hope of decreasing the quantity of coal imported into Canada.

The oil-shales in the Maritime Provinces are mostly found associated with the coal-seams, or lying not far away, and good oil-shales are not much in evidence in the provinces that possess no coal, so that they will be developed relatively to the availability of coal, rather than fostered by the complete absence of coal in their vicinity; and quite probably by a natural extension of the operations of the existing coal companies, who will turn their attention to oil-shale distillation when the scarcity of coal, and consequent high prices, warrants utilization of carbonaceous substances of inferior richness to coal. These substances will include not only the true oil-shales, but carbonaceous shales and low-grade coals, all of which will prove profitable to distill at some future date.

# PROGRESSIVE COAL MINING.[1]

By Geo. A. McHattie.

Annual Western Meeting, Vancouver, November, 1922.

The history of mining is practically the history of civilization. Coal as a mineral was known for centuries before its real value, or its great need in the march of industrial progress, was apparent. Gradually, through the channels of evolution, the coal industry was born. As we understand it to-day, it is a comparatively modern industry, which, we may say, is rather an effect of civilization than a cause. In this respect coal is different from other minerals. Most of these latter — or at least their metallic contents — are an end in themselves, but coal is largely a means to an end. In the progress of modern industry, coal took on an importance not dreamed of in earlier years, and to-day it is undoubtedly the world's most indispensable mineral substance. It is so essential to modern industrial and domestic life that the fear of exhaustion of coal reserves, or even of a probable coal shortage, is sufficient to precipitate a national crisis. It was truly one of the prime factors in the successful culmination of the World War.

It can, therefore, readily be seen that coal is one of mankind's most priceless possessions, and that the mining of coal is one of our greatest basic industries. Indeed, it has become apparent that nations are great or small by virtue of the extent of their mineral lands, and particularly insofar as these have been developed. And in the realm of minerals and industry, coal is king.

What of the origin of the tremendous beds of coal with which our country has been endowed? How come they? In this connection we are told that, at one time, they were considered to be of the Creation. It has only been by slow degrees that this theory was superceded by the more rational one, that coal, as a formation, came by a process of evolution down through the ages. That coal is decayed vegetable matter that has been compressed into the state and shape

[1] Prize Paper, President's Competition, 1922.

which the seams now present, is to-day universally accepted. The quality of the coal found in any particular locality depends, among other factors, upon the character of the original vegetable matter as well as upon the geological age of the deposit.

It is not necessary, however, in this paper to make more than passing reference to the origin and formation of the immense and valuable seams of coal which we have as part of our country's heritage. It is enough merely to draw attention to this great gift, and to pass on to a consideration of the extent and development of our coal seams. What is our national share of this very valuable mineral, the producer of so much wealth?

We find Canada comparatively rich in coal reserves. From an economic standpoint, however, we cannot say their location is of the best. Our known coal-fields are found at the extreme east, on the Atlantic sea-board, in Nova Scotia; a very small quantity in New Brunswick; in the middle west, in Saskatchewan; very extensive areas in Alberta; and in British Columbia, at the extreme west, on the Pacific coast. What of the great extent of territory lying between a gulf as it were? In this section, about two thousand miles wide, are located the great provinces of Quebec, Ontario, and Manitoba. So far as is at present known, they are absolutely barren of marketable coal. Here we have the very backbone of industrial Canada, and not a pound of coal occurring within its limits. Economically, these provinces would be at a great disadvantage, and — particularly in the case of Ontario — they never would have become the thriving hives of industry they are to-day were it not for one of the plans of the Creator, whereby, immediately south of the Great Lakes, in the United States, there was formed what in all probability is the greatest coal-field in the world. By reason of this coal-field, though it lies entirely without the borders of Canada, an economic advantage materializes in favour of the Canadian provinces, which point will be dealt with at greater length later on in this paper. Suffice it to say here that coal, the local lack of which would seem to be such a detriment to the industrial progress of these provinces, comes to their doors in such quantities and at such a price that they are actually independent of, and practically uninterested in, the development of the great Canadian fields naturally their heritage as ours.

We come now to the consideration of mining our coal, and it is here that progressiveness is so badly needed, and so earnestly desired, in the industry.

Progressive methods must be adopted and put into general practice if the industry is to survive the present crisis, and in order that the highest and most economical development may be obtained in our coal districts.

The mining of coal calls for the most progressive engineering ability obtainable, in order that the industry may take on a new lease of life, fully in keeping with its importance in the civil and industrial world. The industry must progress, else stagnation and its resultant evils will overawe the whole situation. A real crisis is facing coal-mining to-day. The public are thoroughly aroused. They are demanding of the coal industry that it play the game, and they cannot be denied. Very recently, United States coals were being marketed in Nova Scotia and New Brunswick while our local mines were practically idle and the miners walking the streets. Facts such as these must receive, and engage, the earnest attention of those under whose direction the management of the mines is placed. The whole industry from the Atlantic to the Pacific needs the injection of a very powerful stimulant. This, in the form of progressive ideas, will do much to overcome many of its ills and ailments.

## THE PROSPECTOR.

Back of the production of any mineral stands the prospector. The searching after minerals is a record of romantic and thrilling episodes in the lives of individuals and of nations. The prospector has always preceded civilized society, following the lure of the mineral. It is often by the merest chance that outcrops and other surface indications of coal are found, so that to be of any value, the prospector must needs be a man possessing considerable knowledge and experience. Canada has yet large tracts of unexplored mineral lands. Therefore a modern progressive policy in coal mining cannot neglect to follow up the work of the great pioneers, who are now almost forgotten in the march of progress, albeit throughout the coalfields of the Dominion the great producing mines are industrial monuments to their memories.

## DEVELOPMENT

With the discovery of coal the prospector passes on, and the work of developing the prospect now engages the attention of those under whose direction the opening of the mine properly comes — the mining engineers. It is at this stage that real progressiveness in engineering ability is needed, to avoid the pit-falls of disaster into which so many good prospects have fallen. A short-sighted policy on the part of those directing the early operations can soon nullify the latent possibilities of the best prospect.

So many things enter into calculations relative to the development of a seam of coal. There is the extent of the deposit, available tonnage, location or position of the seams, the contour of the surface, the chemical composition and physical characters of the coal, and, what is of prime importance, the available market for the product. All these factors must be thoroughly studied and considered ere much development work be done.

Without unduly stressing, at this stage, the importance of progressive engineering ideas, it cannot too strongly be emphasized that much of the later success attained by a mining company is attributable to the solid and far-sighted policy of the first plans of development and operation.

Under this heading, and closely related to plans of development, comes the question of power and equipment. For years, many mining companies took little thought of these prime factors in mining. Fuel was held to be cheap because it was available, and hence any old boiler, engine, or air-compressor would suffice, at least for the temporary installation. How often, oh, how often, was this the beginning of the end. The wreckers were in charge; they lacked progressive ideas, and with direful consequences in all too many cases.

How highly respected are these questions to-day. Power and equipment are receiving attention unknown a few years ago. The electrical and mechanical engineers share with the mining engineer the load of early development. What matters the loss of a little coal? A ton of coal saved is a ton of coal sold. Instead of boiler plants consuming, as formerly, up to 30% of the output of the mine, we find the up-to-date plant burning nothing but refuse. Nothing saleable remains unsold.

The refuse of the mine, being used in the boiler plant, is now handled at a profit to the company. A few years ago steam was taken underground with disastrous results to the equipment of the roadways. To-day we find even that so-called heaven-sent power — compressed air — almost going into the discard with it. What is the reason? Simply that the modern power (electricity) is very rapidly coming to the front and displacing all other forms of power. It is the only progressive power to-day. Its efficiency is 100% — the goal of engineers. It does not damage the mine timbers as does steam; it does not lead to wasteful abuses such as are common with compressed air; it is, in fact, the only truly economic power. The progressive coal mine must have electric power available, to be used wherever and whenever it can be.

## ORGANIZATION.

What of organization around a coal mine? Progressiveness can do wonders here, for success or failure are written in large type across the recording pages when the organization methods followed by the coal companies are studied. We are all familiar with the industrial wrecks entirely due to inefficient and inadequate organization. Give it time, it will ruin almost any prospect, however bright and encouraging the beginning.

The personnel of a coal mine must be an efficient machine above and below ground, or troubles will arise on every side very early. And yet, in and around nearly every coal mine, evidences of inefficient organization and incompetent officials are all too apparent. It is not to be wondered that coal mining in so many districts is facing a crisis. A strong and efficient organization, loyally co-operating for the success of the undertaking, is most essential in every progressive coal mine.

The World War furnished plenty of evidence of the necessity of 100% organization. Throughout the length and breadth of Great Britain for several years nearly everything and everybody was under the influence of one or more branches of the war organizations. The results, as we are all happy to remember, were as encouraging as they were effective, culminating in final victory. This would not have been obtained without efficient organization wide as the world.

In many of the Canadian coal-fields, and particularly in Nova Scotia, local conditions are such that the question of

efficient organization assumes the highest importance. Some of the natural obstacles which have to be faced are high mining costs, inferior coals, extraordinarily thick contiguous seams, liability of spontaneous combustion, and a lack of local markets. In the face of modern competition, these disabilities loom large, and can only be met and overcome with progressive organization.

In this connection, also, a word or two about progressive cost accounting will not be amiss. It is essential that the mine executive officials keep a careful daily check on the mine costs. This enables them, in a most simple manner, to watch closely all the intricate items entering into the daily routine of coal mining, as well as almost instantly to spot leaks or extraordinary happenings throughout the mine or district. These can immediately be followed up and rectified.

Coal mining is a peculiar type of industry, and this largely reacts against its being modernized. However, much can be done by the introduction of progressive ideas and methods. Probably no other industry is so badly hampered with so many hindrances in the way of the out-worn traditions of our forefathers. Mine officials daily have to face these drawbacks underground, and it adds not a little to their already heavy responsibilities. These things must go, and the modern coal mine will arise and take its natural place with the other efficient basic industries of our Dominion.

Just a word in connection with the actual mine organization. What an important official a mine manager is to a company if he is the right man. I suppose in no other industry is there a position carrying with it the responsibilities and actual importance of the colliery manager. We are all familiar with the position and its countless duties and we know that, in any coal mine organization, the pit boss should be the real live-wire of the whole staff if it is to be effective. An inefficient mine manager will soon bring stagnation and disorder about his ears, affecting the whole mine organization. Therefore, in any progressive coal mine, the organization, to be effective, must have an able and progressive mine manager in charge.

A certain wealthy and successful American summed up his success somewhat as follows: My thrills come from success in building up a great commercial organization — one that

is of economic benefit to the nation. It must render the last word in service to those for whom it exists. It must render such service with the greatest possible economy, which means that efficiency must be developed to the *n*-th power.

## OPERATION

Operation is, of course linked up with organization and an efficient organization will operate successfully.

Successful operation entails considerable ability, and is one of the hardest problems facing industry to-day. A sailor at sea contends with the elements natural to that calling. In the operation of a coal mine other natural elements are daily faced, such as outbursts of gas, heavy falls, creeps, disarranged ventilation, and many others, including explosions and fires, which fortunately are not of such frequent occurrence as those enumerated first. But all these things must be faced, and the successful operation is the one that through efficiency in organization meets all these "ghosts of the mine" and overcomes them successfully.

The mining of coal and the handling of it through all the different stages in preparing it for the market is a work of considerable magnitude. The cry of the age is for cheap fuel. The average consumer knows nothing of the trials and dangers of the miner's calling, or of the endless and heavy duties of the mine official. Hence the cry, and the endless talk of coal substitutes. Of course, coal as a commodity has a certain maximum economic value. If the coal cannot be put on the market at a cost in keeping with the value, and in competition with other coals, then it is time to look into the cost of operation. Progressive methods of mining and handling the coal should be introduced. Labour-saving machinery and power, if available and suitable, can do wonders in assisting and bringing new life into a mine, where obsolete methods were causing failure.

Every pound of fuel mined should be put on the market as quickly and cheaply as possible. The coal should be of a good grade and in good condition. The handling should be reduced to a minimum, and thus lessen the breakage so damaging to many of the soft Canadian coals. It is not possible, of course, to avoid trouble with the consumer in this connection,

but the progressive operator can do wonders by the introduction of modern ideas at the coal face, and throughout the passage of the coal to the market. There is far too much reckless shooting and handling on the part of the miner. An excess of powder may save him a little extra picking, but it spells trouble for the operator in the end. The consumer may be wrong in demanding a cheaper fuel in face of rising mine costs, but at least he is sound in his demand for a fuel of a quality and condition in keeping with the selling price.

## THE HUMAN ELEMENT.

No paper on coal mining, and more especially on progressive coal mining, would be complete without devoting considerable space to a discussion on the human element in coal mining. Due to economic, and largely post-war conditions, the question, of labour has become a most acute one. Coal mining, along with practically every important industry, is faced with the great problem of meeting, in a progressive manner, the situation which has arisen. The old order of things certainly does not now prevail. Since the Great War, and taking advantage of every element in his favour, the labourer of to-day is, in his disposition, a hard problem to solve.

In this connection, I suppose no class of labour the world over is harder to handle than the pit-worker. The nature of the work he faces from day to day in the thick darkness of the mine certainly does not tend to the production of an angelic disposition. Rather the miner is prone to dis-satisfaction at his position in the order of things. He usually relishes, and thrives on, complaints, which readily lead to rows. This situation anyone who has had dealings with him knows only too well. A great deal of diplomacy is needed in handling the mine worker.

But we must not be too hard on him. Coal being an important commodity, it naturally follows that the coal miner has only to be told of the important niche he occupies in the industrial world, and you have unrest immediately. Give him a fair deal. The mine is not the most pleasant place in the world to work in, although it has many advantages frankly admitted by those who labour there. The influence on the

temperament of a beautiful clear day in June is missing, and there you are. The miner's pathway is beset with too much shadow. Cheer him up.

What can be done? I say much can be done by the coal mining companies to elevate the pit worker to a higher conception of his place in the industrial world, as an equal of other classes of labour. Unfortunately there is a differential existing to-day. In far too many cases a worker underground is not given the chance socially he is entitled to. Coal mining companies as a rule are very lax in every kind of social welfare work. When the shift is over, and the miner goes to his home, which is generally in poor surroundings, there is nothing done to provide amusement, or instruction of a suitable character tending to make the miner and his family believe they amount to anything. Due to an unfortunate situation existing in most mining districts, the home life is generally inclined to be rough, and it is here that a great deal can be accomplished to lift the workers to a higher plane and give them a broader view of life.

The average mining company leaves its employees entirely to themselves while off duty. Is there not here a great opportunity lost? If the workers' leisure hours were guided along proper paths it is easy to believe the radical labour leaders would not have such rich pastures on which to cast their seed, nor would they be long encouraged in their nefarious calling. A system of social welfare work in the mining communities, fostered and carried on by the mining companies, is certainly one of the great opportunities of the age, and one entirely in line with a progressive policy in coal mining.

## THE MARKETS.

One of the most important departments in any industrial company is that of sales. With the coal mining companies this department does not lose any of its importance, as the question of marketing the product of the mines is ever and always uppermost in the minds of the executive.

In the face of present conditions, progressive methods only will bear fruit in the selling of any commodity, and especially coal. But where are the markets? Naturally one will hasten to say that Canadian coals will always find a ready market in the vicinity of the mines, and from thence as far as economic conditions will permit. Nova Scotia coal apparently

has for its market all the eastern provinces, including Ontario. The production from the prairie provinces can come eastward, and the British Columbia coals can supply the Coast markets. This sounds very well, but we know it does not work out at all in practice. A coal consumption map of the Dominion of Canada will show that practically no Nova Scotia coal is sold west of Montreal, and even in that city the market is not a natural one. We find American coals competing for the Montreal trade, and from thence westward they are in full control of the Canadian market to the borders of Saskatchewan.

What is the reason? Simply economic conditions prevailing which permit American coals to invade the Canadian markets, almost, one might say, without any competition.

The American mines in close vicinity to the Great Lakes have an advantage in the Ontario market which can hardly be off-set in any manner in favour of Nova Scotia coals. During the greater part of the navigation season on the Great Lakes, the great grain carriers, after unloading grain at some eastern point, return west with cargoes of coal. This in turn is loaded into the empty railway cars at the head of the Lakes and transported long distances into the prairie provinces, even against the competition of the Saskatchewan and Alberta coals.

That is an ideal market condition, against which the Canadian companies find it hard — yea, nigh impossible — to compete in the territory concerned.

Referring more particularly to Nova Scotia coals, the argument can be used that the output does not permit of the cry for more markets. Given the Maritime and Quebec business, there is nothing left for Ontario. This argument assumes greater force in view of the possibility of stimulating a greater local demand in the Maritimes by the simple expedient of encouraging an influx of industries in the mining centres.

An ideal condition of business exists when the export of the commodity being produced in any district is compensated by the influx of some other commodity in demand. For both rail or water shipments, it is distinctly to the advantage of coal companies to try and bring about this condition in the mining districts — cars or vessels bringing into the district a commodity not produced there, and taking out coal to other districts, thus establishing a natural ebb and flow of the products of our factories and mines.

In considering the question of markets for our Canadian coals, the purposes to which they are best adapted must be a prime consideration. In most of the Canadian cities where anthracite is available, it is much preferred to bituminous as a household coal. This is natural when the coals are compared. The anthracite is imported into Canada from the United States, since none is found in Canada east of Alberta. Therefore, in this trade in Eastern Canada, the people prefer to import what they want instead of using bituminous coal as a household fuel. American anthracite has also a considerable market in the Maritime Provinces in competition with the product of the lcoal mines.

The idea should be, throughout Canada, to stimulate a greater activity in manufactures. Large boiler plants can burn coal, especially bituminous, much more economically than can the small house furnaces. Central heating plants serving many houses or apartments, such as are maintained in cities like Montreal, should be the arrangement of the future. A market for different grades of Canadian coals might thus be found in competition with the higher grade American anthracite.

In the main, however, the question of cost is what primarily affects the consumer. To get any market, or to hold one in competition, the value of the commodity to the consumer is the crux of the whole situation. Very little sentiment is attached to buying and selling. The value must be there, to get business. If the cost price at the mine is too high, and the ultimate price to the consumer is beyond a reasonable one, there is a very good chance for a competitor to get the market, whether he be Canadian or American.

The Canadian companies are operating in the face of unusually high mining costs, due, largely, to local conditions of a geological nature — or it may be a case of inferior coal. Whatever the cause, it is commonly known that coal mining in Canada is not in a very healthy state. Let us hope that the introduction of progressive ideas and methods throughout the whole of Canada will bring about a state of affairs, in all the mining districts, such that, if the term "progressive coal mining" is in question, we can well exclaim, what other kind of coal mining is there?

# COAL MINING METHODS IN THE CROWSNEST COALFIELD.

By Robert Strachan.

Annual Meeting, British Columbian Division, Vancouver, February, 1922

The method of mining coal or any mineral deposit depends greatly on its form and relative value, and on the nature as well as the extent of the deposit. These considerations lead to the adoption of many different methods of working, even in the same locality, the object in every case being to extract the greatest amount of mineral from a given space, at the lowest possible cost. Where the mineral exists in unlimited quantities, and is of little intrinsic value, its extraction at a minimum cost is of first importance and the loss of a certain proportion of the material is of no consequence. The form to be given to the excavation depends more or less on the degree of coherence of the deposit, the nature of the ground which encloses it, the thickness of the seam, and the angle at which it lies. In view of these and other circumstances, it will be well to give a brief description of the Crowsnest coalfield before dealing with the methods of mining.

The Crowsnest coalfield, which has been described by Dr. D. B. Dowling, of the Canadian Geological Survey, as being, probably, the most important coalfield on the Pacific slope of the Continent, is only one portion of the field on the British Columbian side of the Rocky mountains, the other portions being known as the Upper Elk, the Crown Mountain, and the Flathead fields. Throughout the greater part of this field there have been found twenty-three seams of coal, having

an aggregate thickness of over 200 feet: and the following estimate of the probable amount of coal available here is based on the half of this total thickness of coal.

| | Area | Tons. |
|---|---|---|
| Crowsnest coalfield..... | 230 sq. miles | 25,436,160,000 |
| Upper Elk " ..... | 120 " " | 12,142,147,000 |
| Crown Mountain " ..... | 2 " " | 200,000,000 |
| Flathead " ..... | 6 " " | 600,000,000 |
| Total..... | | 38,378,313,000 |

The nature of the coal varies from bituminous to almost an anthracite. Some idea of the quality of these coals can be gained by referring to the tests made in 1910 at McGill University, at the instance of the Dominion Government, and described by Messrs. Porter and Durley under the title of "An Investigation of the Coals of Canada." From these tests it is evident that the coals from the Crowsnest coalfield compare very favourably with best in the Dominion. Of all the coals examined, only two, namely the Phalen and Harbour seams of Cape Breton, showed greater heating value. In this connection, the following particulars, taken from the Report, are of interest:

| | | | | |
|---|---|---|---|---|
| Cape Breton coals..... | 13,200 | to | 14,010 | B.T.U. |
| Vancouver Island coals.... | 12,470 | to | 13,160 | " |
| Crowsnest coals..... | 13,090 | to | 13,990 | " |

Typical analyses of coals from these districts are as follows:

| District | Moisture | Volatile Matter | Fixed Carbon | Ash | Sulphur |
|---|---|---|---|---|---|
| Cape Breton..... | 3.3% | 33.00% | 54.00% | 7.20% | 2.5% |
| Vancouver Island... | 1.6% | 40.00% | 47.50% | 10.00% | 0.7% |
| Crowsnest..... | 2.0% | 24.00% | 63.50% | 10.00% | 0.5% |

In the Crowsnest field the nature of the overburden above the coal differs greatly with the locality. In some places it consists merely of the surface of timber and a slight covering of coal blossom, while in other places the seams run under the mountains, until the cover reaches almost 3,000 feet.

The coal is easily reached along the outcrop, where it is exposed by the erosion caused by the creeks which traverse the measures, and it can easily be traced along the escarpment on the east side of the Elk river, where the seams can be seen at an elevation of 1,500 to 2,000 feet above the level of the river, dipping into the mountains. Here the great dynamic forces, which have caused the upheaval resulting in the formation of the Rocky mountains, have twisted and folded the measures, making the working of the coal seams along this outcrop difficult and dangerous.

The points at which mining has been successfully conducted are situated where the creeks have cut through the coal measures, exposing them about three to five miles from the Elk escarpment, at, respectively, Coal Creek, Michel, Morrissey, Hosmer, and Corbing. At the present time, collieries at the first two and the last of these localities are working.

The Crow's Nest Pass Coal Company control the largest area in the coalfield, their holdings amounting to about 200,000 acres. At present they are operating only two collieries, namely, Coal Creek and Michel, their once active colliery at Morrissey having been closed down for some years.

## COAL CREEK COLLIERY

Coal Creek colliery was opened in 1897, and is situated about five miles east of the town of Fernie, on the creek after which the colliery is named. Here the coal measures, dipping to the east, reach the level of the creek and are found exposed

on both its north and south side. The seams that have been mined are:—

No. "B" seam, averaging about 5 feet in thickness.
No. 1 " " " 30 " " "
No. 2 " " " 7 " " "
No. 5 " " " 12 " " "
No. 4 " " " 40 " " "

At the present time, three mines are working in the No. 1 seam, three in the No. 2 seam, and one in the B seam.

The coal of No. 1 seam is dark grey in colour, and is very friable. A section would show a hard shale roof, then three feet of coal, two feet of shaly matter, ten feet of coal, three feet of shaly matter, ten feet of coal, more shale and then more coal. A seam, or succession of beds, such as this is very difficult to work, more especially as the shale is very friable, as a result of which it is apt to fall from the roof and sides and it is then very difficult to pick it out from the good coal. The present method is to timber up this shale and also the topmost 3-ft. bed of coal, and to confine mining operations to the upper of the two 10-ft. beds. Needless to state, this method requires not only very close, but very careful, timbering. Timbers are set every three feet, or rather at 3-ft. centres, and closely lagged. Each set, consisting of two posts and crossbar, costs from $1.45 to $2.18, according to the diameter of the timber, making a very appreciable addition to the cost of mining the coal.

The No. 2 seam is rather different from the No. 1. It is thinner, averaging 7 ft. The coal is black and lustrous, and is not so friable as that of No. 1 seam. There is a firm roof of either shale or sandstone, and the floor is composed of shale and coal mixed. This has frequently to be taken up to provide height for haulage. Under normal conditions, there is very little inferior coal or foreign matter in this seam, and the firm nature of the roof permits of lighter timbering and also makes conditions easier for the workmen.

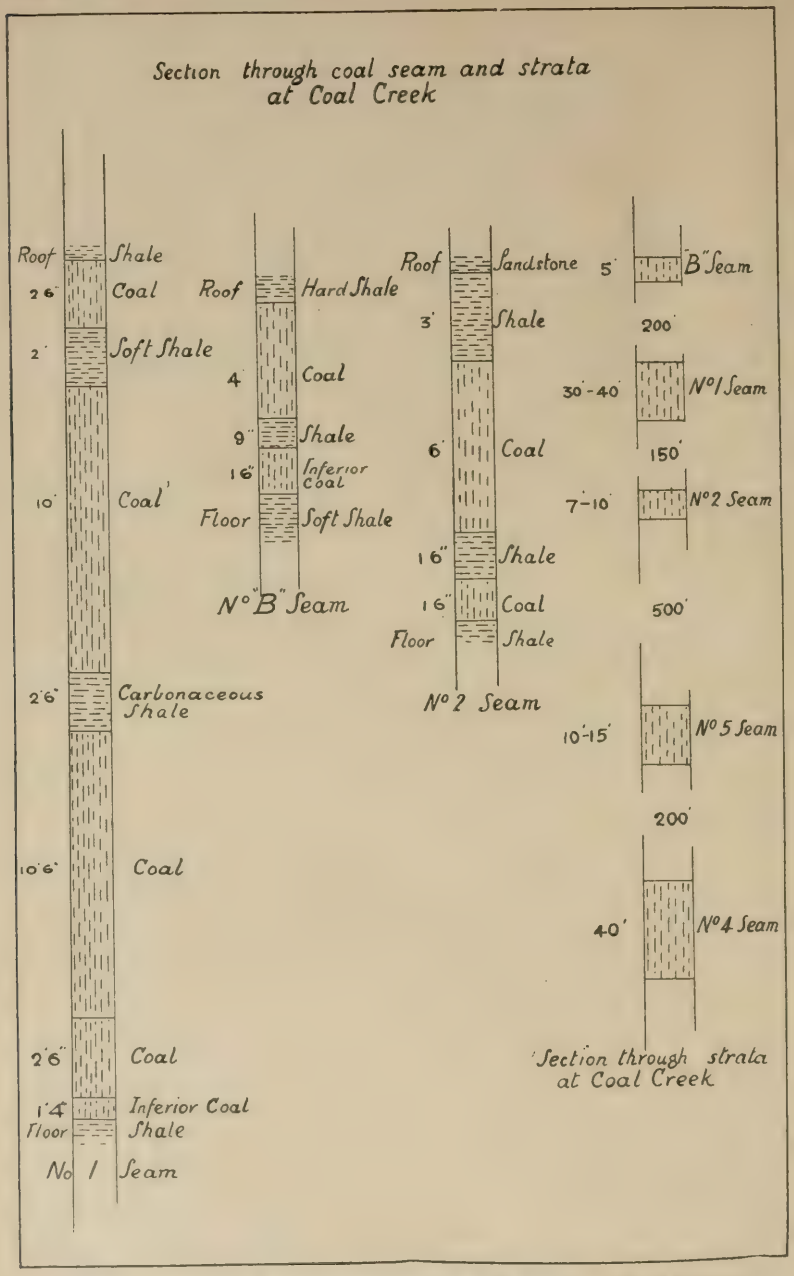


Fig. 1—Section through coal seams and strata at Coal Creek.

The "B" seam averages about 5 ft. in thickness. The coal has a dull grey colour, and is inclined to be very hard. It has a good hard roof of shale, and the floor is an inferior grade of coal, from 2.5 to 3 ft. thick. This material is taken up to provide height for the haulage, and is used exclusively at the boilers for raising steam.

Practically all the coal at Coal Creek is mined by hand, the percentage mined by machinery in 1920 amounting to only 0.057% of the total output. Explosives are not used in getting down the coal.

The coal being easily reached along the outcrop, tunnels were driven in on the seam, level course or at right angles to the dip, and at sufficient height above the creek to allow height for the tipple and other plant necessary to the development of the mines. The method of working the coal is, to some extent, determined by the thickness of the seam, but in every case some form of 'Pillar and Stall' system has been adopted. A 'pillar' is the column of solid coal left in the mine, while a 'stall' is the space or area, from which the coal has been extracted, surrounding the pillar. These pillars were generally 30 by 40 ft., while of the stalls, those intended for main roadways were driven 12 ft. wide, and the remainder 18 ft. wide. In the early days, owing no doubt to the irregular contour of the seam, the pillars in many cases fell short of these dimensions, and little attempt was made to extract any of them.

Several attempts were made to work the No. 2 seam by the longwall method, but these were eventually abandoned for several reasons — mainly the lack of suitable material for packs, and the great thickness of the seam, which necessitated very heavy cogs — which combined to render the method too costly.

At Coal Creek the overburden is at its greatest thickness, and partly because of this, and partly because of the irregular size of the pillars, a considerable amount of trouble has been experienced in working the mines on the south side of the creek. This trouble was occasioned by the strata settling down, thereby breaking the supporting timbers, and, in the case of No. 2 seam, pushing the pillars into the floor, accom-

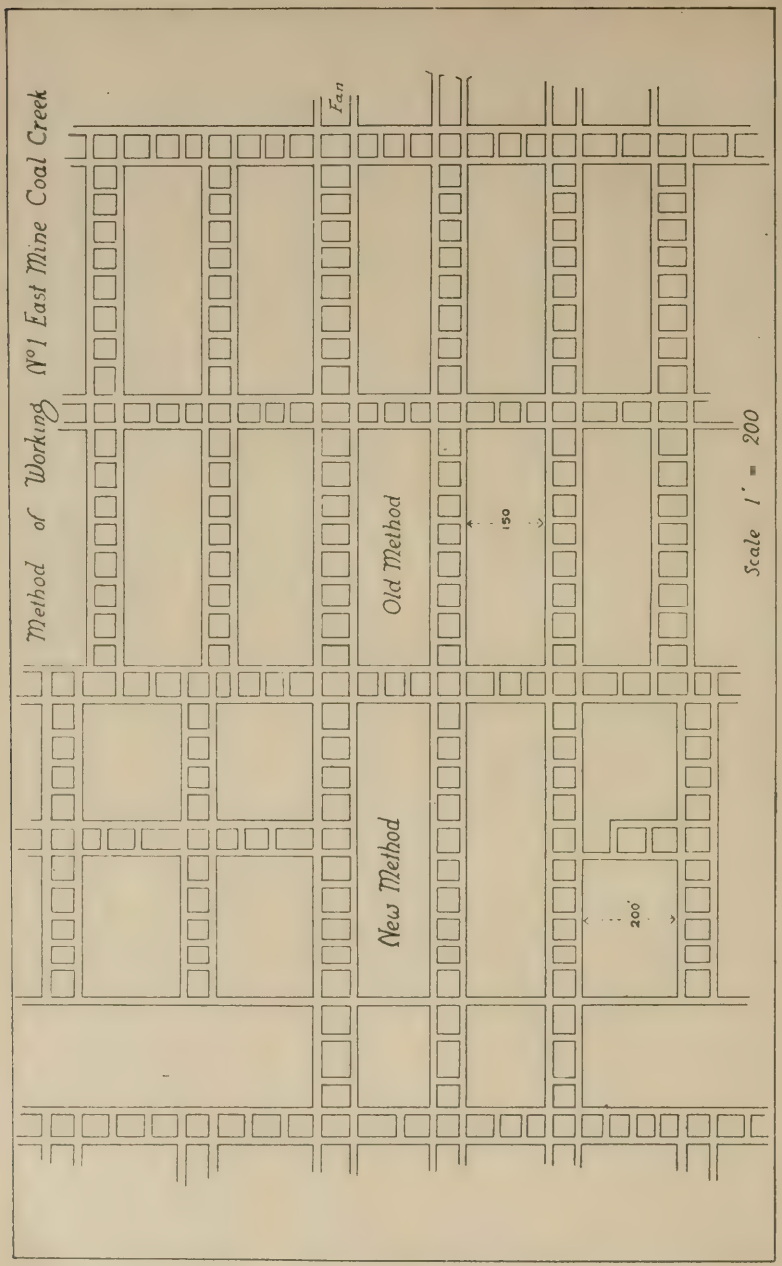


Fig. 2—Method of working No. 1 East Mine, Coal Creek.

panying which was an outflow of explosive gas or methane; and in the case of No. 1 seam causing the sides of the pillars to flow out into the roadways, which in some instances they almost filled. Disturbance of this nature, which is termed locally a 'bump', was very pronounced in 1908, causing the loss of several lives and practically cutting off the greater portion of No. 2 mine. Profiting by the experience in the past, Mr. W. R. Wilson, who had just previously been appointed General Manager for the Crow's Nest Pass Coal Company, determined, in 1911, when laying out the new No. 1 east mine on the No. 1 seam, to increase the size of all the pillars, and to drive his main roadways so that the greatest advantage should be taken of leaving long and solid pillars to assist the ventilation. Following this plan, three pairs of parallel levels were driven, having between each level a 50-foot pillar and between each pair a 150-foot pillar. Butt entries were set off every 600 feet, so that between each pair of entries there was a pillar 380 by 140 ft., which only provided for the extraction, in the first operation, of about 19% of the coal. The development of this mine had proceeded for more than five years, and the output of coal had reached over 1,500 tons per day, when another disturbance, similar to those experienced during 1908 in the No. 2 mine, occurred, practically cutting off the greater part of the mine. This disturbance occurred in the main level district, about 4,000 feet from the mine entrance, and almost exactly over or above the ground where the trouble was manifested in No. 2 mine. The existence of the old workings may have been a contributing cause of the disturbance, but the principal cause was no doubt the great weight of overhead strata, accompanied by possible weakness in the strata. Above this coal seam, the mountain attains a height of over 2,500 feet, which, under normal conditions, represents a weight of 187 tons per square foot; but the conditions here are far from normal, the coal seams being entirely unsupported on the north side, while deep gashes cut into the mountains on both the east and west sides—so that, practically the workings are supported at one end only.

After this disturbance, Mr. Geo. S. Rice, chief mining engineer of the United States Bureau of Mines, was invited to make a report on these 'bumps'; and subsequently a consul-

tation between Mr. Rice, the officials of the Crow's Nest Pass Coal Company, and officials representing the Department of Mines, was held, at which it was decided that, though the six-entry system might safely be maintained as formerly, all future work should provide for still larger pillars. These were to be increased from 150 to 200 ft., thereby reducing the percentage of coal obtained during the first operation from 19% to about 12%. In the other mines, this system is modified to suit local requirements, the size of the pillars being increased from 25% to 30%.

The coal is mined by hand, and loaded into cars having a capacity of 1.7 tons. These are built of wood with iron fittings, have closed ends, and run on a 15-inch Hadfield wheel. The track, of 36-inch gauge, is laid on timber ties; at the coal face the rails are 18-lb., and on the main haulage roads, 36-lb.

In the early days horse-haulage was employed, and coal is still hauled by horses from the face to the main partings. From the partings, however, the coal is handled either by compressed air hoists, by endless rope with bogie attachments, or by compressed air locomotives. All the coal is brought to a common tipple, (built of steel, after the destruction by fire in 1904 of the early timber-built tipple), which stretches right across the valley, for a distance of 780 feet, at a height of 35 feet above the ground. The tipple proper consists of two units, each of which can be operated independently, all the operations being controlled by separate motors. The loaded cars, as delivered to the tipple, are picked up by a car haul, a stop being made at the weigh-scales, where the weight of coal in each car is noted and credited to the miner whose check is attached to the car. The loaded car runs by gravity into a circular dump, displacing the empty car, and automatically locking itself. The dump makes a complete revolution before coming to rest, and unloads the coal to a feed which regulates the supply to the shaking screens, where the lump is separated from the fines. The lump passes from the screen to a picking table, where the foreign material is picked out, before it is loaded into cars for shipment. Provision is made for loading into either open or box cars. In the case of the former, a long swing loading boom is used to reduce breakages, and a Smith's gravity box car loader is provided for dealing with

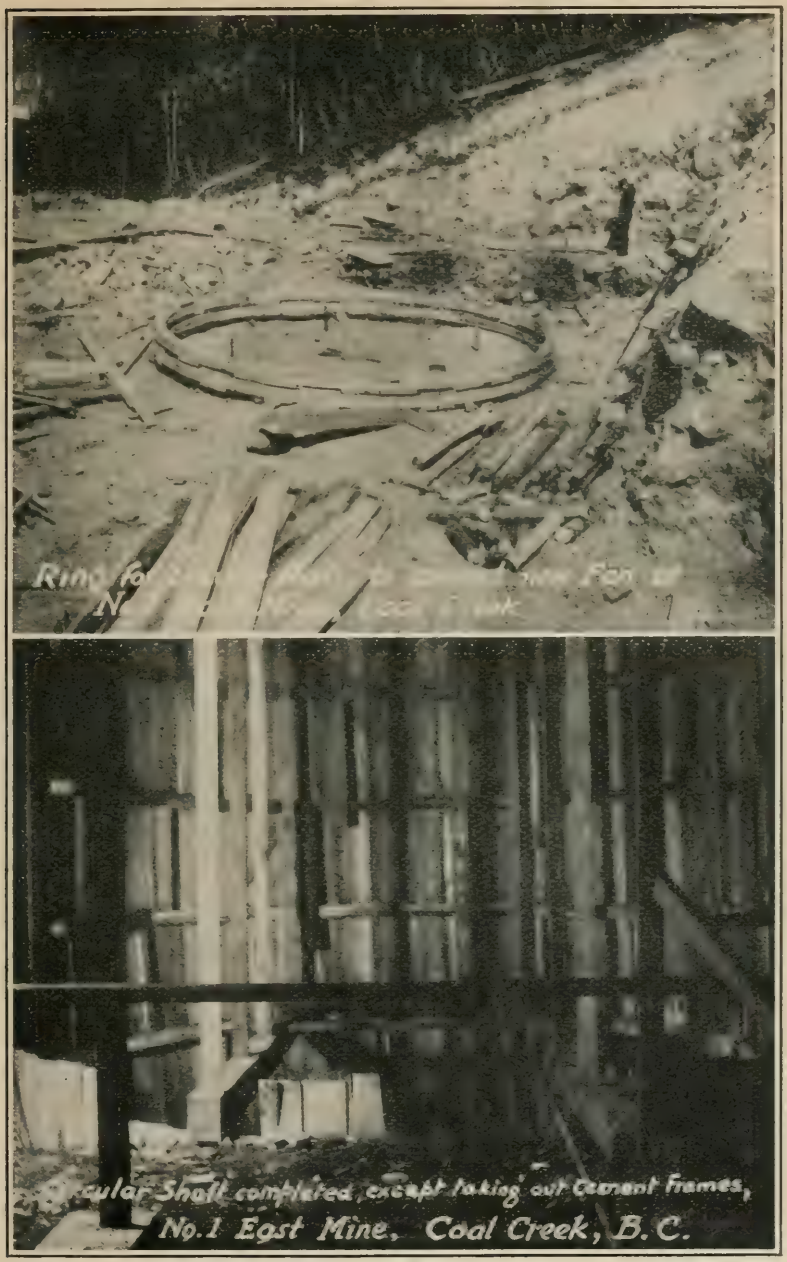
Circular Shaft completed, except taking out Cement Frames,
No. 1 East Mine, Coal Creek, B. C.

Fig. 3

the latter. The slack passing through the screens is taken by a belt conveyor to a hopper, from which it is drawn, as required, into open cars. Formerly this slack or fines was used in the coke ovens at Fernie, but as these have not been in operation for over two years, the slack is now marketed.

To ventilate the mines and to carry away dangerous or noxious gases, large quantities of air are required, to supply which each mine is equipped with a ventilating fan. The fans at the larger mines are of the steam-driven Wilson type; at the smaller mines they are of the electric-driven Keith type. The Wilson type fan is capable of producing up to 150,000 cubic feet of air per minute: the maximum capacity of the Keith is 60,000 cubic feet per minute. The "Coal Mines Regulation Act" requires the provision of at least 100 cubic feet of air for every man in the mine; at the Coal Creek mines the supply is greatly in excess of this specification, as the following table shows. That this should be so is necessary on account of the exceptionally great amount of methane given off in these mines.

In the following table, each man or boy is reckoned as one unit and each horse or mule as three units. The table shows (1) the amount of air provided per unit per minute, (2) the amount of methane produced per ton of coal mined, and (3) the amount of methane given off every twenty-four hours:—

| Mine | Amount of air circulated per minute for each unit. | Methane produced per ton of coal mined. | Methane given off every 24 hours. |
|---|---|---|---|
| No. 1 East | 1,000 cu. ft. | 6,192 cu. ft. | 2,476,800 cu. ft. |
| No. 3 | 624 " | 2,697 " | 868.320 " |
| No. 2 | 476 " | 1,360 " | 345,600 " |
| No. B North | 552 " | 3,086 " | 610,560 " |
| No. 1 South | 331 " | 1,867 " | 672,480 " |
| No. 1 North | 268 " | 384 " | 102,240 " |
| No. 9 | 460 " | .... " | 82,080 " |
| Total | | | 5,158,080 " |

It will be noted that in the mines with the greatest production of methane or explosive gas, the greatest amount of air per unit is provided. This is as it should be, and the

Fig. 4

new fan which has been built for the most gaseous mine, No. 1 East, will almost double the present supply of air per unit.

The amount of methane given off per ton of coal mined, or every twenty-four hours, may seem startling. The figures nevertheless represent averages for the whole of the year 1920. However, the amount of air circulated in the twenty-four hours reaches 444,859,200 cubic feet, so that the average percentage of methane is merely 1.2, which is far below the ignition point for an explosive mixture, unless coal dust be present.

Dangers from coal dust are guarded against by treating all the roadways, sides, roof, and floor with water. Where available locally, the water is kept under pressure by means of a pump driven by compressed air; otherwise it is piped in. In some portions of the main roadway, sprays are arranged to blow off at every 100 yards, all night, and in this manner the moisture permeates every portion of the roadway. In addition, flue dust and second burnt ashes are in use as a preventive to the propagation of an explosion should such unfortunately occur. In spite of all these precautions accidents are liable to happen, and within the past four years all oil-burning lamps (except those required by the officials, not exceeding twenty), have been eliminated from the mine in favour of the Edison electric mine safety lamp. The use of safety lamps of the oil-burning type by the mine officials is permitted in order that they may be able to make easy and rapid determinations of the mine air, as to the presence of gas or methane; and although some may criticize the decision to prohibit the carrying of an oil lamp by the miner, thereby depriving him of a means of detecting the presence of dangerous gas, I am convinced that, with the provision of regular inspections, the maximum of safety is ensured by the present system. In addition to testing for gas by the safety lamp, determinations are made with the Burrell gas detector at least once in every eight hours in every mine. Records are made of these readings, and a diagram, kept in the manager's office, shows the daily percentage of methane given off, the barometric pressure, the amount of air passing at such

Coal Creek Colliery and village, Coal Creek, B.C.
Weathered outcrop of coal in ravine, Coal Creek, B.C.

Fig. 5

times as it is measured, and the results of the analyses of the air samples taken. In this way a reminder of the constant danger due to the presence of gas is kept prominently in evidence. Any unusual rise in the methane percentage is noted, and any decrease in the atmospheric pressure, which generally portends an abnormal outflow of gas, can be watched and prepared for.

The management of the colliery is under the sole control of the manager, who has an assistant.Each mine is in charge of an overman; and each district is supervised by a fireboss. All of these officials hold certificates under the "Coal Mines Regulation Act".

The present source of power is steam (although in the near future electric power will be supplied from the Alberta & British Columbia's power plant in course of construction at Bull river). The boiler equipment consists of fourteen tubular boilers, with a capacity of 3,000 horse power Three low pressure compressors, having a combined capacity of 6,000 cubic feet of free air per minute compressed to 80 lb. pressure, supply power for hoists, and for pumping inside the mine. One high pressure compressor, having a capacity of 1,346 cubic feet of free air per minute compressed to 900 lb., supplies power to operate the compressed air locomotives used for haulage. Three generators, producing 680 amperes at 250 volts, supply power for the motors on the tipple, for driving the fan, and for the lighting of the plant as well as of the houses of the workmen residing at Coal Creek.

Stable accommodation is provided for over one hundred horses, which are required for haulage purposes inside and outside the mine. There are also adequate machine shops, carpenter shops, and car-repair shops, as well as the usual offices, and warehouse accommodation. A large well-kept lamp-room is maintained where all the workmen's lamps are re-charged if electric, and, if oil, are filled and cleaned previous to their issue. In the wash- and change-house are over 680 lockers for the convenience of the workers.

A majority of the workmen reside in Fernie, five miles distant. There are, however, over 200 workmen's houses at Coal Creek; and an efficient train service is maintained between Fernie and Coal Creek to suit the workers.

## MICHEL COLLIERY.

Michel Colliery, opened in 1899, is situated about 17 miles north of Fernie, and 8 miles west of the summit of the Rocky Mountains. Here Michel creek has cut across the coal measures leaving several seams exposed on either side. The seams that have been worked are:—

| | | |
|---|---|---|
| No. 3 seam | 5½ | ft. thick. |
| No. 3 upper seam | 12 | " " |
| No. 4 seam | 10 | " " |
| No. 5 " | 8 | " " |
| No. 7 " | 8 | " " |
| No. 8 " | 12 | " " |
| No. 9 " | 14 | " " |

At present three mines are working, two on the No. 3 upper seam, and the third, No. 8, on the No. 8 seam. The No. 3 upper seam averages about twelve feet thick, and has two partings running through it. One of these partings is very narrow, being under one inch; the other varies in thickness from one foot to four or five feet in some parts of the field. The coal is dark with a bright lustre, similar to that of the No. 2 seam of Coal Creek, and is of a firm texture. The roof and floor are of hard shale. The No. 8 seam ranges from 10 to 30 feet in thickness. The coal is of a dull grey colour, similar to that of the No. 1 seam at Coal Creek.

In working the No. 3 upper seam the timbering problem is easy; but in No. 8, owing to the varying thickness of the seam, it is necessary sometimes to leave much coal overhead, and this makes the timbering more difficult, as in the No. 1 seam at Coal Creek. Costs and methods of timbering are very similar to those at Coal Creek.

In locating the coal, the same procedure was followed as at Coal Creek; but since at Michel the creek cuts across the measures, instead of directly against them as at Coal Creek, the seams on the one side of the valley rise with the mountain, and on the other side dip directly into it.

After the seams were proved, a cross-cut tunnel was driven cutting through Nos. 5 and 4 and into No. 3 seam; and since 1917 this tunnel has been extended so as to cut the No. 3 upper seam, part of which had been developed through a short tunnel from the No. 3. No trouble has been experienced

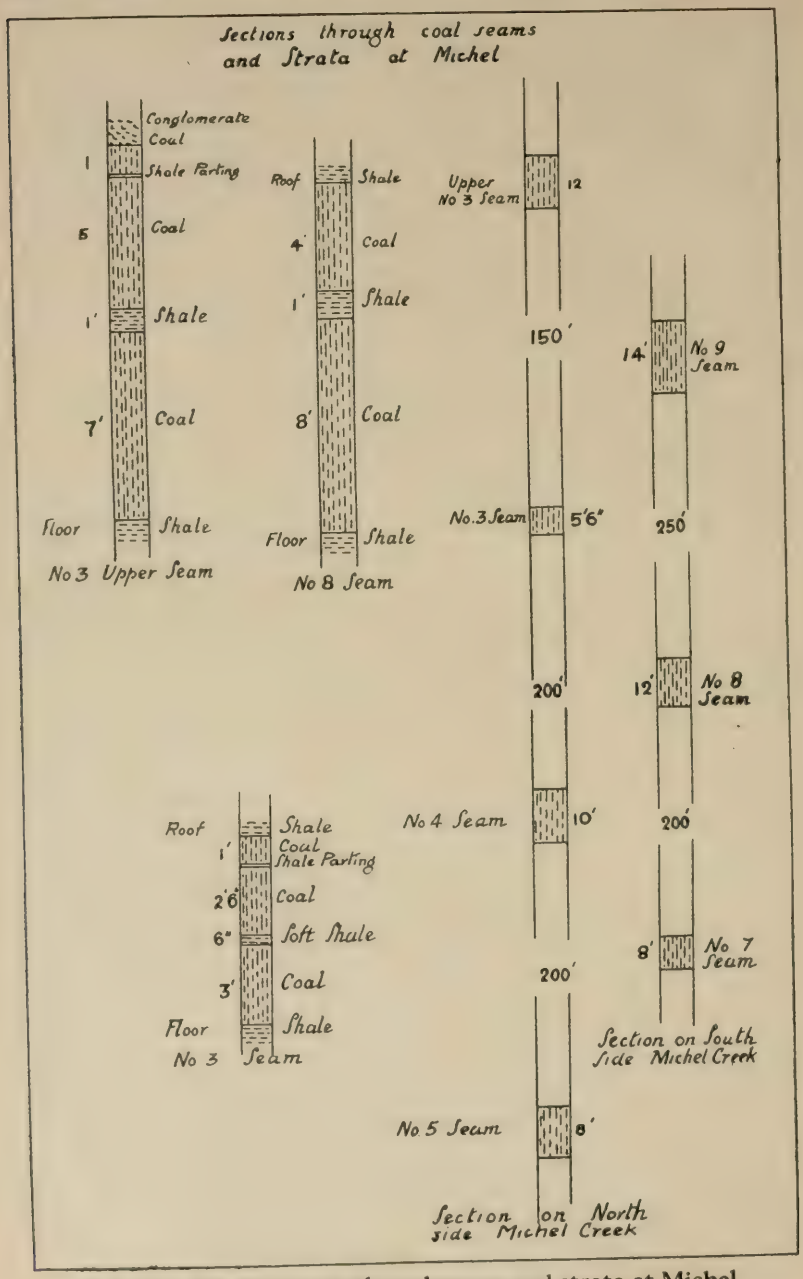


Fig. 6.— Sections through coal seams and strata at Michel.

with 'bumps' or disturbances such as have occurred at Coal Creek; and although on the south side we may attribute this immunity to the lightness of the cover of overhead strata, on the north, where the cover reaches almost 1,500 feet, the reason seems to be the greater stability of the formation, which is there practically supported on all sides.

In all the mines at Michel explosives are used, to a limited extent, for blasting down the coal, and the proportion of machine-mined coal is greater than at Coal Creek, although not more than 4.3% of the total output.

The cars used here are slightly different from those at Coal Creek, being shorter. Some are built of timber and iron, and others entirely of iron. The wooden cars have a capacity of 1.3 tons; the iron cars carry 1.6 tons. The track gauge here is 42 inches, instead of 36 inches as at Coal Creek.

The procedure as to the loading of cars, lighting, and supervision is very similar to that at Coal Creek.

The No. 8 mine, which is situated about 400 feet above the tipple, has a small dumping and storage arrangement, whereby the mine cars are unloaded and returned into the mine. The coal is drawn from this storage into a skip with a capacity of 8 tons, which lowers it to another storage pocket where the skip unloads automatically. From this storage the coal is drawn into mine cars and taken to the tipple by an endless rope. The unloading and loading of the coal would ordinarily appear to be detrimental by causing much breakage, but here this is not an important consideration as practically all of this coal goes to the coke ovens. The tipple, which is of the Green patent double track type, is a steel structure, and is about 600 feet long. On the south side, where coal from the No. 8 seam, is handled, the cars are placed on a hydraulic swing dump which first tilts the cars up to an angle of between 60 and 70 degrees, and when the door is open allows the coal to run out of the car to the feed belt which supplies the screen. The hydraulic transfer then drops down, making a connection with a lower or under track, and enables the empty car to run to a point exactly beneath the position it formerly occupied on the upper track. In this manner the hydraulic transfer not only dumps the loaded car, but transfers it from the upper to the lower track. On the north side, or No. 3, and No. 3 east side, the loaded

Michel Colliery, Michel, B.C.,
Coke Ovens at Michel Colliery, Michel, B.C.,

Fig. 7

cars are delivered by compressed air locomotives to the foot of an incline leading to the tipple, where they are picked up by a Green's patent car haul, consisting of a double endless chain arrangement, travelling on car wheels connected with axles, which act as spreader bars, as pushers, and as retarders of the mine cars when on the incline. The tracks for the mine are inside those for the chain haul, and when the cars are delivered to the chain-haul the cross-bar moves up to the rear car wheels and pushes the car up the incline to the dump, which is simply a continuation of the track increasing to an angle of from 60 to 70 degrees. At this angle, when the car door is open, the coal runs out into the feed belt which supplies the screens, and the car, continuing to travel with the chain haul, is pushed into a swing lift transfer, while the cross-bar, following the sprocket wheel around which the chain haul travels, carries the rear end of the car with it over the top of the sprocket to an overhead track, by which the empty car returns down the incline, the cross-bar now acting as a retarder.

The tipple is fully equipped with shaking screens for separating the slack from the lump coal, picking and handling it into either open or box cars similar to the practice at Coal Creek. The slack is taken by conveyor and bucket elevator to the slack bins, from which it is drawn as required for the coke ovens. All the machinery around the tipple is driven by electric motors. Ventilation for the mines is provided by a fan at each mine. At No. 3 East is a steam driven Wilson fan having a capacity of 150,000 cubic feet per minute; at No. 3 mine, the fan is a steam driven Murphy, having a capacity of 130,000 cubic feet per minute; and at No. 8 is a Keith fan, driven by compressed air, and having a capacity of 60,000 cubic feet per minute. The following tables show (1) the amount of air produced per unit per minute, allowing, as in the case of Coal Creek, one unit for each man or boy and three units for each horse or mule, (2) the amount of methane produced per ton of coal mined, and (3), the amount of methane in every twenty-four hours

| Mine | Amount of air circulated per minute for each unit. | Methane produced per ton of coal mined | Methane given off every 24 hours. |
|---|---|---|---|
| No. 8 | 271 cu. ft. | 211 cu. ft. | 95,040 cu. ft. |
| No. 3 East | 1,009 " | 6,096 " | 1,828,800 " |
| No. 3 | 551 " | 1,732 " | 606,240 " |
| Total | | | 2,530,080 " |

To dilute this amount of dangerous gas, 254,822,400 cubic feet of air is circulated every twenty-four hours, and the methane percentage averages just under one per cent. Similar measures to those at Coal Creek are adopted to deal with the coal dust problem, men being detailed in every mine to keep the roof, floor, sides, and working face free from dust.

The power plant consists of eleven tubular boilers, having a combined capacity of 1,600 horse power. Two low-pressure compressors having a combined capacity of 8,000 cubic feet of free air per minute, compressed to 80 lb. per square foot, provide power for pumps, hoists, and the driving of No. 8 mine ventilating fan. One high-pressure compressor, having a capacity of 1,450 cubic feet of free air per minute compressed to 900 lb., supplies the power for the compressed air locomotives. Two generators, of 250 K.W. each, supply power for the motors connected with the tipple, and for the lighting of the plant and the town of Michel. All the buildings around the power plant are of brick, as is also the lamp-room where the electric lamps used by the miners and the oil lamps used by the officials are kept in good repair. Other buildings include well equipped machine and carpenter shops, offices and warehouse, and a commodious wash and change room, fitted with both hand-basins and showers.

The coal company own practically all the houses in Michel, where the majority of the workmen reside.

## CORBIN COLLIERY.

Corbin Colliery, situated at the extreme east of the province and about 20 miles directly east of Fernie, was opened in 1908 by the Corbin Coal and Coke Company, whose holdings comprise about 20,000 acres. Here the structures which were caused by the forces that formed the Rocky Mountains are of such a nature that this portion of the coalfield, locally known as Coal Mountain, has been almost entirely detached from the rest of the field.

In the surrounding mountains, the coal formation can be seen, well up towards their summits, lying at about the inclination that is common throughout the field; but here, at Coal Mountain, the seams have an almost vertical attitude, and they attain great thicknesses. This latter feature appears to

be due to monoclinal folding of the measures, and in a horizontal section of 2,604 feet we have 814 feet of coal. The whole of the mountain has not yet been prospected, and although over a million tons of coal have been extracted since the colliery was opened, this represents no more than a scratch on the surface.

The first seam opened here was No. 1, which had a thickness of 170 feet. As the seam was developed it was found to split into two benches, one 70 and the other 100 feet thick, the first being called 'No. 1 Left', the other 'No. 1 Right.' This seam was worked by driving four parallel levels on the same horizontal plane. These were connected by cross-cuts and raise-ups every 40 feet, and when the raise was up 40 feet rooms were turned off, making pillars 40 feet square — a 'Pillar and Stall' method applied to vertical seams. When this work had reached the outcrop, about 1,600 feet from the level, the extraction of pillars was begun, but before this had proceeded very far fire among the loose coal caused trouble, and occasioned the closing down of the mine.

Prior to the abandonment of No. 1 mine, work had been commenced on a large deposit of coal known as No. 3 seam about one mile south and at about 1,200 feet higher elevation—practically 6,200 feet above sea level. Here the cut reveals a synclinal basin about 370 feet in width, having near the centre a thickness of coal amounting to 200 feet. This deposit is in a direct line with the seams at Corbin, and although the geological relations have not yet been established, drill holes and open cuts would seem to indicate that this is a continuation of the same bodies of coal.

The method of mining this coal in the first instance consisted of stripping off the timber, then washing away the surface, which consisted of coal blossom, and loading the coal directly into the railway cars with a steam shovel. But owing to complaints of the pollution of the streams, this method had to be abandoned and steam shovels were then used for stripping the overburden, which was loaded into dump cars, removed some distance from the workings, and dumped.

On account of the high altitude of this open cut, difficulty was experienced in building a railway up to it; four switchbacks were required. Then, during the winter, owing to the

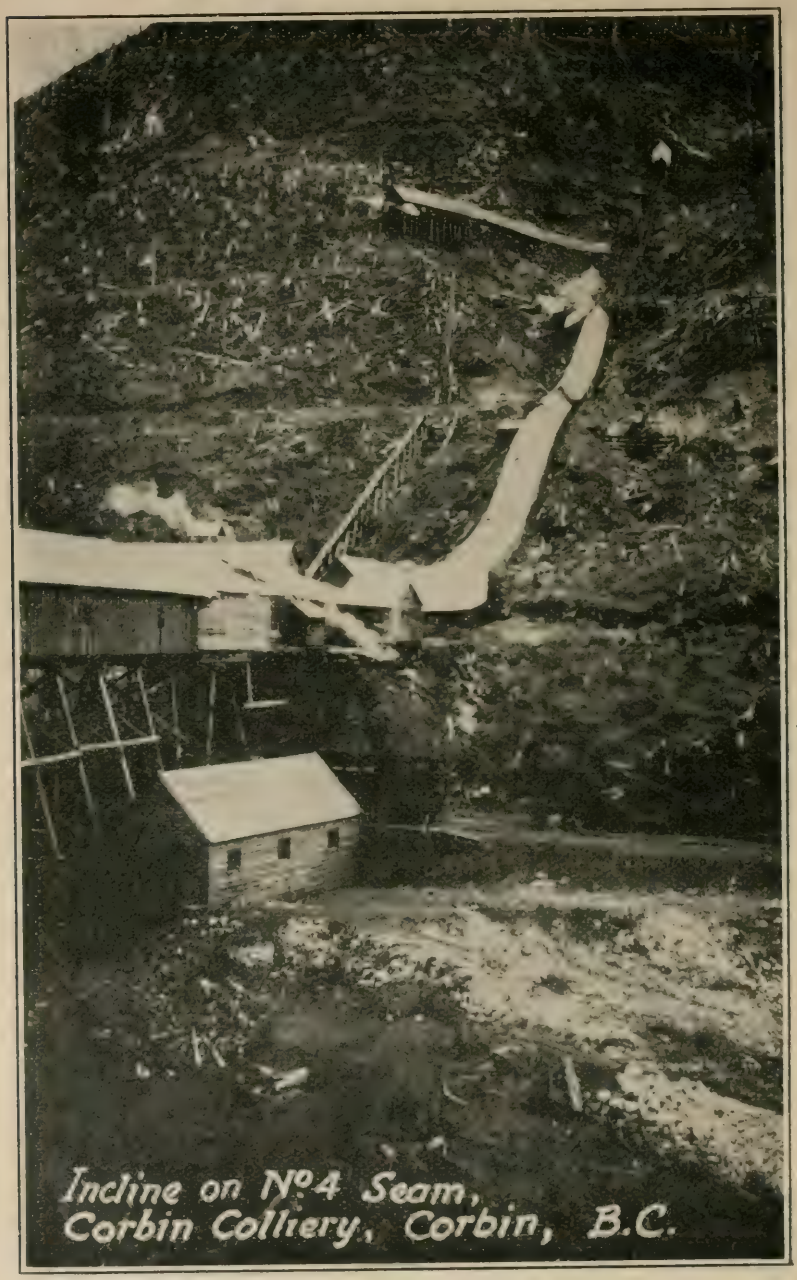


Fig. 8

heavy snow-fall, trouble was experienced in keeping the tracks clear; but the workings being on a side-hill, this difficulty was finally overcome by installing turn-tables at the end of each switch, to turn the snow plough and locomotive.

The mining of the coal by the stripping method did not prove as economical as might have been expected, and the amount of coal which was lost in the stripping operation was considerable. Had the operations been planned on a less spectacular scale, with provision for more benches, smaller cars, and for chutes or retarding conveyors to take the coal down the hill, during which operation the coal could have been prepared for the market, better results would no doubt have been obtained.

The No. 4 seam was also opened about this time, and as was the case in the No. 1 seam, it was found to be divided into two portions by an intrusive body of rock, one portion measuring 50 feet thick, the other 150 feet. This seam was developed along similar lines to the No. 1, but, in the extraction of pillars, the caving system was practised. In this system the pillars are first formed and then split into small portions, when, owing to the weight either of the surrounding ground or of themselves, the coal runs of its own accord. In the event of the coal failing to run, however, either one of the existing cross-cuts is used or a new one is driven to the point and a blast set off. Owing to the friable nature of the coal here, no trouble was experienced, unless timber which had not been withdrawn caused a stoppage. In this manner practically the whole series of pillars can be drawn until all the coal is extracted; but unless the roof or floor is also of a tender nature and follows up the coal, trouble is likely to be experienced with the huge cavities left behind.

Seams Nos. 2 and 5 have also been opened, the first to the west of Nos. 1 and 4, the other to the east; but so far they have only revealed a thickness of twelve feet of coal, and very little development has been carried out on either of them.

The latest seam to be opened at Corbin is the No. 6, which is situated about 1,200 feet east of the No. 1 seam. Here the coal has been proved to be 420 feet thick. Three tunnels have been driven in on it at different levels, and the measures seem to be more regular here than elsewhere in this field.

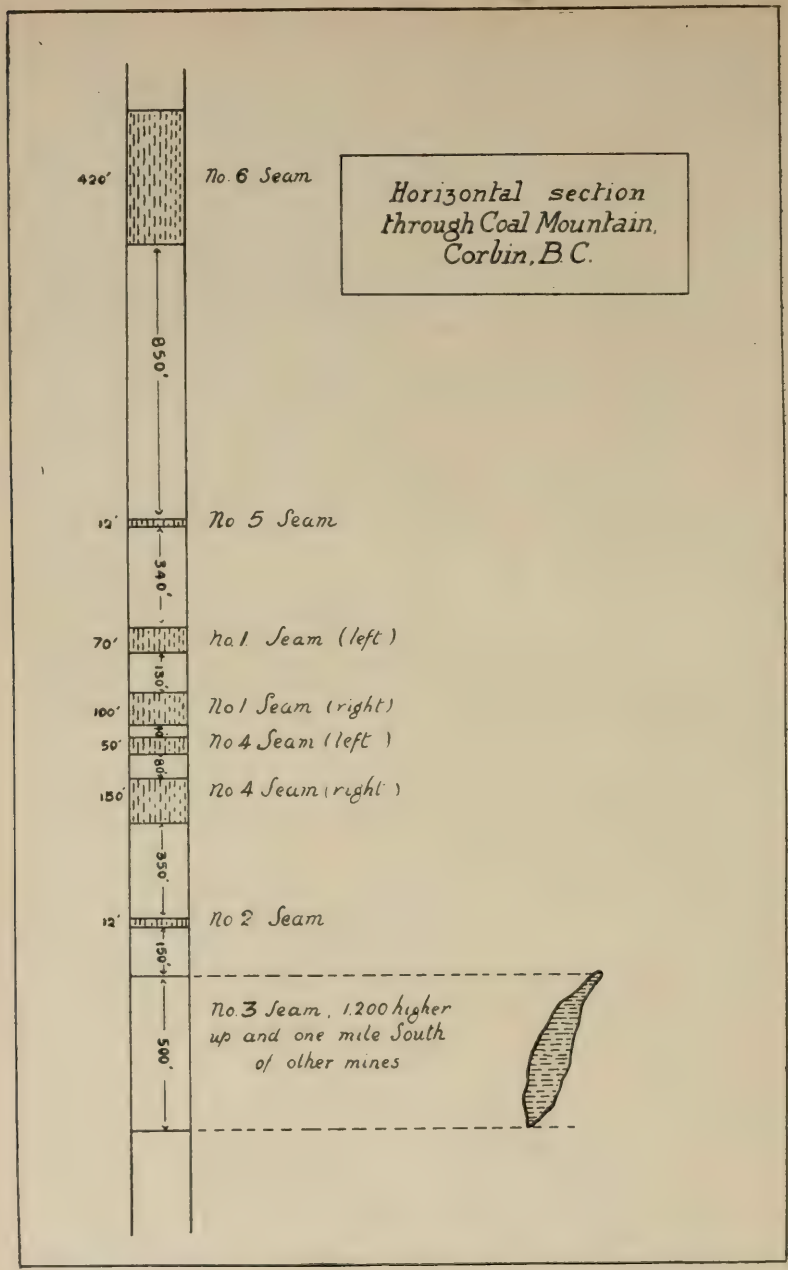


Fig. 9—Horizontal section through Coal Mountain, Corbin.

In the development of this mine a new plan will be tried, somewhat similar to that in use at the iron-ore mines in Michigan. The seam being laid off in panels, raises are driven up until the size of the panel is reached, and the pillars will then be extracted in lifts. A solid floor of timber will be laid down under the tracks and this will act as the roof for the next lift, and so on.

One factor that has seriously retarded the development of these large seams of coal has been their liability to take fire. In addition to the No. 1 seam being on fire, the upper portion of the No. 4 is also burning, and only by the strenuous efforts of the officials at Corbin has this mine been saved so far. Whether or not these fires are caused by spontaneous combustion has not been difinitely determined. It is suspected that the first fire in the No. 1 seam was communicated from the outside by a bush fire; and owing to the thinness of the strata between Nos. 1 and 4 seams (only 40 feet), and to the fact that a great many pillars of coal have been extracted in the former and cogs of timber built in their place, there is a suspicion also that the fire in No. 1 seam may have spread to No. 4 through pre-existing breaks, or breaks caused by the excessive heat generated by the fire.

Thus the seams which have been mined at Corbin are:—

| | | | |
|---|---|---|---|
| No. 1 seam | ..... | 170 feet | thick. |
| No. 2 " | ..... | 12 " | " |
| No. 3 " | ..... | 200 " | " |
| No. 4 " | ..... | 200 " | " |
| No. 5 " | ..... | 12 " | " |
| No. 6 " | ..... | 420 " | " |

In all the seams, the character ot the coal is essentially the same. In appearance it is black and glossy, and analyses show that it possesses a greater ratio of free carbon and less of volatile matter than the coal either of Michel or of Coal Creek, compared with which it is also much higher in ash. The results of tests to determine the coking qualities of the coals have not, so far, been favourable.

In consequence of the almost vertical position of the seams, chutes are generally used to bring the coal down to one level, where it is received into cars, of about two tons capacity, running on a 42-inch gauge track. Blasting is carried on to

Fig. 10

a limited extent only, and no machines—other than the steam shovel, which has not been in use during the past year — are employed. The haulage is by horse to the tipple, where storage bins are provided to hold over one thousand tons. A Marcus screen is also installed, so that the coal can be screened and sized if necessary. The present arrangement is for the coal to be loaded into railway open cars at Nos. 5, 6, and 3, and either shipped as run-of-mine coal or sent to the Marcus screen to be sized and picked. If the latter, it is switched into the storage pit track, where it is unloaded and taken by conveyor to the Marcus as required.

Timbering, lighting, and supervision provisions are practically the same as at Coal Creek.

Owing to the vertical pitch of the seams the question of ventilation is easily solved. At the No. 4 mine a small Keith steam-driven fan practically ventilates all the levels; and at No. 6 a small Sirocco fan driven by a motor provides the ventilation. On the same unit basis as for Coal Creek and Michel we have:—

| Mine | Amount of air circulated per minute for each unit. | Methane produced per ton of coal mined. | Methane given off every 24 hours. |
|---|---|---|---|
| No. 4..... | 700 cu. ft. | 136 cu. ft. | 37,440 cu. ft. |

To deal with this quantity of methane there is 41,040,000 cubic feet of air circulating, so that the average percentage of methane is less than 0.1%, and there is practically no coal dust, all the roads, sides, roof, and floor being damp and in some cases practically running with water.

The power plant consists of three boilers of the return tubular type, their total capacity being 275 horse power. One 70 H.P. electric generator, alternating type, 2,300 volts, affords the power for driving the small fan at No. 6 mine, for the machine shop, and for lighting the works and the town of Corbin. A machine shop, carpenter shop, offices, and warehouse are provided, as well as a lamp room and wash house.

## HOSMER AND MORRISSEY COLLIERIES.

In addition to the three collieries above described, two others have been in operation but are now closed. These are

the Hosmer colliery, belonging to the Canadian Pacific Railway Natural Resource Department, and closed in 1914, and the Morrissey colliery belonging to the Crow's Nest Pass Coal Co., and closed in 1909.

At Hosmer, ten seams of coal were cross-cut by the Main tunnel, which is about 5,000 feet long. The principal cause for the abandonment of this colliery was, it is understood, the troubled nature of the ground, which was along the Elk escarpment and consisted of five square miles of coal lands and one square mile of surface. In these coal lands, which parallel the Elk river, the measures have been twisted and folded.

In the Morrissey Colliery, great trouble was experienced from blow-outs of gas, mostly methane, entailing a considerable loss of life. The coal, moreover, did nor prove very suitable for coking purposes, and was so very fine that it did not make a suitable fuel for locomotives. On these accounts the Crow's Nest Pass Coal Company decided to concentrate effort for the meantime on the operation of the other two collieries at Michel and Coal Creek.

## GENERAL NOTES.

Michel, Coal Creek, and Corbin have rail connection with the Canadian Pacific system, the first two by the Morrissey, Fernie and Michel Railway, the latter by the British Columbia Eastern. Coal Creek and Michel have rail connection with the Great Northern system, over which a great percentage of the coal from these collieries passes to the United States.

That the coalfield is of immense value is evident when it is realized that over 17,000,000 tons of coal have been won from it since it was first opened in 1897, and that almost 4,000,000 tons of coke have been made from the coal produced. On a very conservative estimate there still remains in the Crowsnest coalfield alone 1,500 times this amount, and if we include the Upper Elk, Crown Mountain, and Flathead areas, the amount of available coal is more than 2,250 times the tonnage extracted up to the present.

The market for the coal in the early days comprised the Canadian Pacific Railway for steam purposes, the Prairie districts for household use, and the Kootenays for steam and

smelting purposes. The first-named market is now supplied principally from the steam-coal mines on the eastern slope of the Rockies; the second from the lignite mines in the vicinity of that market; and as to the third, the demand for coke has fallen from 250,000 tons in 1905 to less than 70,000 tons in 1920. Under these circumstances it is not surprising that the development of this great coalfield has been retarded.

At the present time almost sixty per cent of the coal produced is exported to the United States, where it finds a ready market in competition with the coals from Wyoming, Washington, and Montana. The natural market in Canada for the coal would seem to be to the west, but existing freight rates and adverse grades are prohibitive restrictions, and the conditions in this respect must be ameliorated before a market in this direction can be developed.

# THE EOCENE COAL BASIN AT CHU CHUA, B. C.

By W. L. Uglow.

Annual General Meeting, British Columbian Division, Vancouver, February, 1922.

It has been known for a great many years that coal occurs on the lands of the Indian Reservation, near Chu Chua, B.C. The exact locality of the occurrence is in the gorge of Newhykulston creek, at a distance of three-quarters of a mile above its mouth. This creek empties into the North Thompson river, 45 miles north of Kamloops.

The district was examined in 1877 by G. M. Dawson. In his report,[1] he gives a detailed stratigraphical section of the coal measures, and correlates them with his Coldwater group of Oligocene age.

## Geology.

A detailed examination of this locality was made by the writer in July, 1921. The exposures of the coal measures are limited to the sides of the valley of Newhykulston creek, and the total thickness occurring in the visible section is about 600 feet.

In the vicinity of Chu Chua, on the east side of the North Thompson river, the country is very largely underlain by greenstone pillow lavas of the Fennell formation. These lavas constitute a series of bold bluffs, rising from the bottom of the valley to the edge of the plateau 3,000 feet higher up. Their age is uncertain, but they are believed to be either Palæozoic or pre-Cambrian. They form a part of Dawson's Adams Lake series. Along and parallel to the base of these bluffs is a low flat-topped ridge, partially timbered and completely covered with glacial drift. The top of this low ridge is about 800 feet above the North Thompson river. It follows along the foot of the valley slope for a distance of five miles. Several mountain streams have cut narrow steep-sided channels

[1]Geol. Surv. Canada, Rep. Prog., 1877-78, pp 112B-114B.

through this ridge, but only on Newhykulston creek are any of the underlying country rocks exposed. These are seen to consist of massive to thin-bedded greyish-white arkosic sandstone, arenaceous shale, graywacke, quartz pebble conglomerate, carbonaceous shale, and several seams of coal. This group of rocks constitutes the Chu Chua formation. The basal conglomerate of the formation is exposed in the Canadian National Railway cuts, a short distance north of Chu Chua station, and is probably 1,500 feet stratigraphically below the lowest exposures on Newhykulston creek.

The coal measures strike N 15° E, parallel to the trend of the North Thompson valley in the vicinity, and they dip 20-25 deg. easterly, or toward the rim of the valley bottom. Newhykulston creek, which flows in a westerly direction, affords a partial cross-section of the measures. The extent of the Chu Chua formation north and south of its outcrops in the creek valley could not be determined on account of the paucity of exposures. The topography of the long low ridge suggests, however, that rocks of this formation underlie its covering of glacial drift throughout its length of five miles.

The Chu Chua formation seems to have been deposited in an Eocene lake which occupied a valley co-extensive with that of the lower portion of the North Thompson river. The sediments, no doubt, extended from rim to rim of the lake and filled it to a depth of many hundred feet. Several coal seams occur within this thickness. After the deposition of these sediments, the country was tilted about 25 degrees to the east, and erosive agencies removed part of the coal measures. Outpourings of Miocene lava (hornblende andesite) covered the eroded surface of the Chu Chua formation. These lavas were responsible for the baking, or induration, of the sediments, and for the raising of the rank of the coal from lignite to subbituminous.

A period of faulting followed that of extrusion, and the tilted sediments and overlying lava-flows were broken by a series of strike faults parallel to the trend of the North Thompson valley. Erosive agencies, acting since the Miocene, have removed large portions of the Tertiary formations, leaving only a few remnants in sheltered re-entrants along the edges of the valley bottom. The Chu Chua coal basin constitutes one of these erosion remnants.

## THE COAL MEASURES.

Within the stratigraphical section of 600 feet of the Chu Chua formation, there are exposed at least 14 coal seams, varying in thickness from 4 inches to 45 inches. Chief among these are the Thomas "A" and "B" seams (No. 1 mine), the Smith "A" and "B" seams, and the Gray seam (No. 3 mine). These are on the property of the Chu Chua Coal Company, the only operators.

Collections of fossil flora were obtained (1) from a thin shale bed adjacent to the Thomas A seam; (2) from a shale bed in a cliff of sandstones and shales of the Chu Chua formation on Joseph creek, 10 miles north of Newhykulston creek; and (3) from shale and sandstone beds exposed in the gorge of Darlington creek on the western side of the North Thompson trench.

These collections were submitted by the Geological Survey to Professor D. W. Berry, of Johns Hopkins University, for determination of stratigraphical position and species [1]. Professor Berry states quite definitely that the fossils are of middle or upper Eocene age. This is interesting in that the Chu Chua basin thereby becomes the first authentic discovery of a commercial coal basin of Eocene age in the interior of British Columbia — commercial in the sense that coal has been shipped from it.

## THE COAL SEAMS.

*The Thomas Seam.*— Until the end of the summer of 1921, development of the Chu Chua basin had been almost entirely confined to the Thomas seam. It is a double seam, the upper part being known as the "A", and the lower as the "B", seam. In the outcrop along the side hill, these two portions of the seam are separated by 9 to 10 feet of arkosic sandstone and arenaceous shale. At the time of the writer's examination, they were seen to be not more than three feet apart in the bottom of the slope. At a still lower level, they may unite and form a single seam four to five feet thick.

[1] Professor Berry's reports are published in detail in the Summary Report of the Geological Survey of Canada, Part A, 1921.

The two portions of the seam vary in physical characteristics and thickness within very short distances along the beds. The following sections will reveal in detail the nature of the seams:

(1)

Sandstone roof

| | | |
|---|---|---|
| Hard black lustrous coal (streak brown) | 12 | in. |
| Sandy shale | 1½ | " |
| Hard lustrous coal | 1 | " |
| Grey sandstone | 3 | " |
| Hard black lustrous coal (streak brown) | 10 | " |
| Sandstone | 3½ | " |
| Coal | 7 | " |
| Sandstone floor | 24 | " |

Total coal: 30 inches.

(2)

Sandstone roof

| | | |
|---|---|---|
| Hard massive lustrous coal | 8 | in. |
| Shale | 2 | " |
| Coal | 1½ | " |
| Shale | 1 | " |
| Coal | 7½ | " |
| Sandy shale | 2 | " |
| Coal | 5 | " |

Sandstone (smooth floor)

Total coal: 22 inches.

(3)

Sandstone roof

| | | |
|---|---|---|
| Coal — Thomas "A" seam | 30 | in. |
| Sandstone | 36 | " |
| Clean coal — Thomas "B" seam | 30 | " |

(1) Vertical section of Thomas "B" seam on south wall at the bottom of main slope.

(2) Vertical section of Thomas "A" seam on north face, lower level.

(3) Vertical section of Thomas "A" and "B" seams, 20 ft. north of No. 1 chute, lower level.

*The Smith Seam.*— This also is a double seam and is exposed in a short slope on the north side of the creek valley. The detailed stratigraphical section at the portal of the slope is as follows:

Massive pebbly sandstone roof

| | |
|---|---|
| Hard, thinly laminated, fairly clean coal | 18-20 in. |
| Hard massive grey sandstone | 20-48 " |
| Shale with coaly partings | 6 " |
| Coal with shale partings | 10 " |

Fissile grey arenaceous shale floor

*The Gray Seam.*— This is the lowest coal so far discovered in the measures. It was uncovered at the time of the stream freshet in June, 1921, by the sapping of a thick mantle of glacial drift. At the suggestion of the writer a small excavation was made at this point so as to determine the thickness and sample the seam. The coal was much weathered even in the face of the excavation, so that it was difficult to ascertain just how much clean coal there might be in the 45 inches of coaly material exposed.

*Physical characteristics and grade of the coal.*—The coal is a hard, black, lustrous, thinly-laminated variety, with a dark brown streak. It is very friable but it does not readily blacken the fingers. The seams lack the perpendicular prismatic jointing so characteristic of bituminous coal, but possess instead the thinly-laminated structure found in lignites. The coal weathers rapidly and becomes coated with a yellowish-white alteration product. It burns readily with a long yellow flame, and it is said to be a good steam-producing fuel.

Small lenses, blebs, and irregular-shaped masses of a semi-transparent, amber-yellow material occur within the coal seams. This substance is the fossilized form of some of the resins of the coal-forming trees. As a result of the bio-chemical and dynamo-chemical changes which took place during the formation of the coal, the woody portions of the seam were much carbonized and fractured, while the resins resisted the bio-chemical changes to the last [1]. Being of a pliable nature, the fossilized resins accommodated themselves by a sort of plastic flow to the shapes of the fractures in the coal, and now they occur filling cracks and cavities of all shapes; and in places they present the appearance of having replaced the coal.

Four samples of the coal were taken from the seams and were submitted to the Mines Branch, Department of Mines,

[1] U.S. Geol. Surv., Prof. Pap. 85, 1914.

Ottawa, for analysis. The samples were analyzed as received (R), and also after drying at 105 deg. centigrade (D). The results are given below:

| | (1) | | (2) | |
|---|---|---|---|---|
| | R | D | R | D |
| Moisture.....per cent | 4.0 | | 3.7 | |
| Ash..... " " | 24.0 | 25.0 | 37.3 | 38.8 |
| Volatile matter..... " " | 36.1 | 37.6 | 29.4 | 30.5 |
| Fixed carbon..... " " | 35.9 | 37.4 | 29.6 | 30.7 |
| Sulphur..... " " | 2.7 | 2.8 | 0.6 | 0.7 |
| Fuel ratio..... | 0.99 | 0.99 | 1.01 | 1.01 |
| B.T.U..... | 10,290 | 10,700 | 8,230 | 8,550 |
| | Sample made poor coke in very small lumps. | | Sample agglomerates | |

| | (3) | | (4) | |
|---|---|---|---|---|
| | R | D | R | D |
| Moisture.....per cent | 3.6 | | 4.0 | |
| Ash..... " " | 13.8 | 14.3 | 22.1 | 23.0 |
| Volatile matter..... " " | 37.9 | 39.3 | 37.9 | 39.5 |
| Fixed carbon..... " " | 44.7 | 46.4 | 36.0 | 37.5 |
| Sulphur..... " " | 0.7 | 0.7 | 0.8 | 0.8 |
| Fuel ratio..... | 1.15 | 1.15 | 0.95 | 0.95 |
| B.T.U..... | 12,040 | 12,490 | 10,780 | 11,230 |
| | Small lump, fair coke. | | Small lump, poor coke | |

(1) Sample represents a channel of 15 inches in length across the Smith "A" seam, at a point 30 feet down the slope from the portal.

(2) Sample represents a channel 41 inches in length across the Gray seam. It includes 4 inches of coal shale, 8 to 10 inches of hard lustrous coal, 1 inch of clay, followed by weathered soft shaly coal to the bottom.

(3) Selected sample from the upper 8-inch portion of the Thomas "A" seam.

(4) Selected sample from the Thomas "B" seam.

These results show that the coal should be classified as low-grade low-rank bituminous, or as low-grade high-rank sub-bituminous.[1]

*Development.*— During the summer of 1921, the development of this coal basin was confined to the Thomas seam, on which there was an adit 125 feet in length, with workings which continued down the slope for a distance of about 200 feet.

[1] U.S. Geol. Surv., Prof. Paper 100-A, 1917, p. 3.

Coal was mined on this seam from beneath 5,400 square yards of surface. This work was discontinued about the end of the year.

During the winter of 1921-22, underground development on the Gray seam was commenced and coal has already been mined here from beneath 13,000 square yards of surface.

A short slope on the Smith seam was driven several years ago and abandoned.

About 25 men were employed by the Chu Chua Coal Company during the summer of 1921.

*Production.*— During September, 1921, the Chu Chua Coal Company was producing from 35 to 80 tons of coal per week. Toward the end of the year, the output increased to about 40 tons per day.

*Extent of Basin.*— As stated above, the longitudinal extent of the Eocene basin near Chu Chua could not be determined on account of the overburden. Judging from the underground development, and from topographic data, it is reasonable to expect that the Chu Chua formation underlies the low ridge which parallels the base of the valley slopes. This should be determined by diamond drilling.

The continuation of the coal seams down the dip will, of course, be cut off where they overlap the surface of the Fennell greenstones, which represent the old shore line of the lake.

A detailed stratigraphical section of the coal measures in Newhykulston creek shows a gap of about 1,500 feet between the Gray seam and the basal conglomerate. In this gap, in which there are no exposures, may be found coal seams of commercial value.

Coal may also be found at other points along the North Thompson or associated valleys, where erosive agencies have not entirely removed the rocks of the Chu Chua formation.

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## DISCUSSION.

MR. J. CUNLIFFE: I have no dispute with Dr. Uglow personally, nor do I object to most of what he said. What I do

take issue with is that he did not say enough, and what he did say was sufficient to leave the impression on his audience that the Chu Chua coal area has little if any merit as a commercial proposition.

The impression left on the audience was well summed up in the remarks of the Chairman, Mr. Thomas Graham, who said that, as far as he could learn from the paper, the Chu Chua seams appeared to be more in the nature of geological specimens than commercial seams — or words to that effect. If that summarises the impression left on a coal man of Mr. Graham's experience, that conveyed to less experienced men may be easily surmised.

If, as I understood, Dr. Uglow stated that the length of the exposure of the coal measures along Coal creek was from 600 to 700 feet, the statement is entirely inaccurate. I made a survey in June, 1918, over the part of the creek along which the exposures extend. Commencing at the easterly or upper end of the exposed measures, at the point where they pass under the surface silts, the measured distance down the stream to the outcrop of No. 1 seam — named in Dr. Uglow's paper the "Thomas" seam — is 628 feet. Continuing down the creek, the outcrop of No. 2 seam was crossed at 1,440 feet, or 812 feet from No. 1; and that of No. 3 seam — named in his paper the "Gray" seam — at 1,490 feet, or 862 feet from the No. 1 seam.

Dr. Uglow gave the analysis of a sample taken from a part of the No. 3 seam at the surface, the material of which was partly covered with clay and gravel, and was little better than a black smudge; and which, until recent development, was considered of no value. His sample may have some interest from a scientific point of view, but it is useless as a fair sample of the seam. Development on this seam was taking place a short distance away, and it is strange to me that he did not go into the mine and get his sample. Mr. McEvoy, on behalf of the Geological Survey, examined this area in 1892, and found the thickest band of coal in the No. 1 seam to be 18 inches; the best Dr. Uglow could find was 8 or 9 inches.

The majority of the audience are laymen, who do not make the proper distinction between the duties of the Government geologist and the mining engineer. The geologist, a Govern-

ment employee, has the reputation of being — and usually is — impartial. Inferences made by him, even in the absence of direct statements, are accepted by laymen in preference to the opinion of the reputedly less impartial mining engineer, when the views of the two do not happen to coincide. The Government geologist is seldom a practical mining engineer, and, although well grounded in theoretical principles, he lacks the training and intuition which can be acquired only by long practical experience. These men are sent out by the Department to make surveys and collect geological data for the districts to which they are assigned, and are not permitted to give advice, or to recommend or condemn, either by innuendo or inference, any property encountered in their work.

The mining engineer frequently follows the geologist, and his work is to ascertain, by exploration and development, whether, in his opinion, the property has a commercial value or not. The duties of the two are as far apart as the poles.

Another small point to which I wish to call your attention is that, in my opinion, Dr. Uglow's plan of the Chu Chua coal basin, as thrown on the screen, is inaccurate. It shows Coal creek — the stream on which the outcrops are exposed — to be approximately the centre of the area. It does not need much geology to tell where the crystalline trap rocks cut off the probable coal bearing sedimentaries, which in this case happens about three-quarters of a mile south of the creek, while the similar northern cut-off is over 3½ miles distant.

I would like to ask Dr. Uglow, at whose request was his paper given at this time? It is a departure from a rule which has been in force by the Department for years. This rule forbids its field men from reading papers or giving lectures before the public on any part of their work before publication in the Annual Report, except with the express permission of the Director.

This is the second attack by Dr. Uglow on this coal area, and I would like to ask him his reason for doing so? His first attack was perhaps excusable to some extent, inasmuch as he believed he was misrepresented by interested parties in an article which appeared in a Kamloops newspaper last summer. But I fail to find either a reason or an excuse for this second attack.

The following is a description of the seams as proved for short distances from the outcrops by underground work. I leave the verdict to practical coal men experienced in the dirty seams operated in western British Columbia.

*Seam No. 1.*— This is the upper and most easterly seam yet discovered in this area. It varies from 3 ft. 6 in. to 3 ft. 9 in. thick, and has running through it two small bands of clay from 2 in. to 3 in. thick each. At 10 feet below No. 1, a seam of good cannel coal, 3 ft. 0 in. thick, is exposed at the surface, clean and free from waste. At 150 feet down a dip slope on No. 1, the layer of strata between the two seams is reduced to one foot in thickness, and the two are worked as one seam. The latest reports state that two thin bands of waste appear in the cannel seam where it is pierced in the slope on No. 1, although none is visible at the surface.

*Seam No. 2.*— This seam is about 300 feet lower in the strata than No. 1, and about 800 feet distant from it on the surface. At the time of my last visit — Feb., 1921 — a dip slope was driven on the seam, and was down about 30 feet. The following is a section of the seam taken at the face by myself in the presence of Mr. David Brown, mine manager at the time:

| | |
|---|---|
| Roof, sandstone, several feet thick | |
| Coal band next to roof | 2 ft. |
| Sandy shale | 2 ft. |
| Bottom coal | 2 ft. |

*Seam No. 3.*— This is the lowest seam yet discovered in the series; it is the one named the "Gray" seam in Dr. Uglow's paper. It is 5 ft. 9 in. thick, of which the bottom 3 ft. is clean coal; the upper 2 ft. 9 in. is a mixture of coal and bone — waste — in about equal parts.

Exploration work at the outcrops of these seams, to prove thickness and quality, has been carried on steadily for the past 18 months. More than 5,000 tons of the coal has been sold for domestic use in Kamloops, and it has been pronounced a first-class fuel. The work, which includes about 1,200 feet of underground driveage, has been done under the supervision of two practical certified coal mine managers of B.C. The area was examined by the late Mr. Alexander Sharp, mining engineer for Mr. Pat Burns for many years; by Mr. James

Ashworth, late general manager of the Crow's Nest Pass Coal Company and a coal mining engineer of international repute; and by several others. All of them are satisfied that the seams are of a workable thickness and quality, and they are of the opinion the seams extend over a sufficient area to make a coal mine of moderate dimensions.

I had daily supervision underground for four years at the No. 4 mine of the Canadian Collieries at Cumberland — Comox — the most extensive mine and the most widely explored seam on Vancouver Island; and I can say without hesitation that with the exception of some thick swellings in the Comox seam, probably about 25% of the area operated, the Nos. 2 and 3 seams at Chu Chua, as far as explored, compare favourably as regards thickness and waste matter with the No. 4 seam at Comox.

PROF. W. L. UGLOW: To the charges made by Mr. Cunliffe, I can only reply that, in my paper, I presented the actual facts of the case, acquired during a detailed examination of the geology in the valley of Newhykulston creek, which occupied six days of my own time and four days of the time of my party. Every available portion of the coal workings was examined, and my samples were taken from the underground working faces, wherever there were such.

I deny the charge of having made two attacks on the coal area. For the first so-called "attack", I refer you to the columns of the Kamloops *Telegram* and *Standard Sentinel* of the early part of August, 1921. (I do not remember the exact dates of the issues). Whether or not the paper I have just presented may be called an "attack", I leave to the audience before me.

Replying to Mr. Cunliffe's statement that my figure of 600 feet for the thickness of the exposed coal measures is incorrect, I can simply state that this figure is the stratigraphical thickness accurately measured from bed to bed. Naturally I was not talking about the length of exposures measured along the stream bed.

Regarding my sample of material from the No. 3 or Gray seam, I refer you to the notes accompanying the analysis given in my paper.

Regarding Mr. Cunliffe's distinction between the business of the Government geologist and the mining engineer, I may

say that it is his privilege to hold that opinion if he so desires. I leave the matter to the meeting.

Regarding the sketch plan of the Chu Chua coal area which I rapidly prepared as a slide, I indicated to the meeting that it was diagrammatic. Had Mr. Cunliffe observed that both vertical and horizontal scales were given, he might not have been so convinced that the sketch was doing an injury to his prospects in the district.

There is no necessity for replying to his question as to what authority I have for presenting my paper before this Institute.

Regarding Mr. Cunliffe's descriptions of the three seams, with their thicknesses, I refer the meeting to the detailed sections I have given of each of these seams. In these sections, I have differentiated carefully between "coal seam" and "coal" (meaning clean coal). You will notice that my sections are given in greater detail than Mr. Cunliffe's.

Regarding No. 3 or the Gray seam, until the time of this meeting neither Mr. Cunliffe nor myself had seen the underground workings on it. I have a letter from the superintendent of the Chu Chua Coal Company to the effect that, at the bottom of the new slope on this seam, there are three small bands of coal with a total thickness of the seam of 5 ft. 4 in. As I did not obtain this information first hand, I did not embody it in the paper.

I deny the truth of Mr. Cunliffe's statement that I did not mention No. 2 or the Smith seam in my paper. I gave a detailed section and sketch of the seam, and the analysis of a sample.

In the presentation of the paper, I emphasized the possibility of the longitudinal extent of the coal basin, as well as the fact that my samples were near-surface ones. I stated definitely that I had been engaged in making a scientific investigation of the Chu Chua coal deposits, and that I was not presenting a report on the commercial possibilities of the district.

The questions as to whether my classification of subbituminous for the coal is correct and as to whether the coal measures are of commercial character, I prefer to leave to the judgment of the large number of coal mining men who are in the audience.

# THE COAL MEASURES OF CUMBERLAND AND VICINITY, VANCOUVER ISLAND.

By J. D. MacKenzie.

Annual Meeting, British Columbian Division, Vancouver, February, 1922.

## Introduction.

It is now more than fifty years since the Geological Survey began its field work in British Columbia. In 1871, A. R. C. Selwyn and James Richardson, coming from "Canada" by way of San Francisco, began the series of investigations which the Survey has carried on continuously since that time. Selwyn's party explored the valley of the North Thompson and crossed into the Upper Fraser valley, which they followed nearly to the Yellowhead pass. Richardson, after spending the summer in the interior of the province, began his work on the coal measures of Vancouver Island in October, and continued this investigation intermittently for five years. Ten years ago the Nanaimo coalfield was examined in detail by C. H. Clapp, then of the Geological Survey, who also made a hasty reconnaissance of the so-called Comox area.

Coal has been mined at Nanaimo since 1852[1] and at Cumberland since 1888[2]. Three-quarters of a century

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(1) Dawson, G. M., Geol. Surv. Can. Ann. Rept., Vol. III, 1887-88 page 80R.

(2) Idem., page 81R.

of mining, together with the investigations of the officers of the Geological Survey and other geologists, have made it evident that many interesting facts bearing on the geology of coal are to be gleaned on Vancouver Island. Some of this evidence has already been published by Richardson, Dawson, Clapp, and others, and the present paper is a contribution to the same subject.

The district described here is in the vicinity of Cumberland. It consists of 80 or 90 square miles lying in a strip 6 to 7 miles wide along the east coast of Vancouver Island, and extending from Brown and Puntledge rivers on the north to Tsable river on the south. In this district the Canadian Collieries (Dunsmuir) Ltd., have done much prospecting, and are operating mines at Cumberland. I am indebted to the generosity of the Canadian Collieries, and especially to Mr. Thomas Graham, general superintendent, for access to valuable information possessed by the company concerning the coal-bearing rocks of Vancouver Island.

It is the intention of this paper to outline the general geology of the district, with special reference to the conditions under which the coal was accumulated.

## SUMMARY.

The district here described is an area of between 80 and 90 square miles, situated on the east coast of Vancouver Island, in the vicinity of the town of Cumberland.

In this area the sedimentary Nanaimo series of Upper Cretaceous age rests unconformably on rocks of the Vancouver group of Lower Jurassic and Triassic age. These latter rocks are metamorphosed and altered effusive and pyroclastic volcanics which were deeply weathered in pre-Upper Cretaceous time.

On their weathered surface, after surmergence, was accumulated the Comox formation of the Nanaimo series, which is dominantly composed of massive, homogenous, white and light grey sandstone, containing coal seams. Locally at its base are found coarse angular breccias formed of fragments of the underlying volcanic rocks, and more rarely a basal conglomerate of a similar nature has been observed. Characteristically, however, the basal measures are fine; sandstone, shale and even boney coal having been observed lying directly on the pre-Cretaceous volcanics. The Comox formation is of variable thickness, ranging from 80 to 1,000 feet and averaging about 600 feet; the variation being principally due to the inequalities of the surface on which it was accumulated. In certain places a band of intraformational conglomerate about 60 feet thick occurs; it is generally conformable, and locally has a thickness of 600 feet. Where thus thickened it is at the expense of the sandstone and coal seams, which it has eroded and replaced.

Five distinct coal seams can be recognized in the Tsable River area. Each seam may consist of several benches of coal separated in almost all cases by shale. In thickness the seams vary from a few inches to 25 feet. Considerable variations in thickness and even in quantity of coal may occur in the same seam in a distance of a few score yards, and even more abrupt variations are known to occur. Almost always the coal is accompanied by a band of brownish grey shale, and frequently shale forms both roof and floor.

The only other member of the Nanaimo series represented in this district is the Trent River formation, which is dominantly composed of shale, though sandstone and an important intercalation of conglomerate also occur; but no trace or indication of coal has been observed. The Trent River formation is about 1,000 feet thick, and, so far as has been ascertained, it is conformable on the Comox formation.

Northwest of Cumberland, a local intrusion of igneous rock has taken place, cutting across, and materially affecting, the coal seams adjacent to it.

The Comox and Trent River formations have a general dip of about five degrees to the northeast, consequent on a regional

tilt in that direction; but occasionally steeper dips are found both to the northeast and southwest, so it is apparent that there has been some folding also. Large breaks are not in evidence, but small tension faults are troublesome in some places in the mines, and seem to be connected genetically with the folding.

In that part of the district known as the Tsable River coalfield, favourable outcrops, supplemented by a large number of boreholes, have made it possible to construct a contoured map of the pre-Cretaceous surface. From this map, after taking into account the degree of irregularity of the old surface caused by the post-Cretaceous folding, the initial irregularities of the ancient terrain have been deciphered, and in this way the manner in which the overlap of the Comox formation took place, and the original extent of the coal seams, has been worked out.

## CONCLUSIONS.

(1). The pre-Upper Cretaceous surface in the area now occupied by the Comox formation was an irregular one, with a total relief of 440 feet at least.

(2). This irregular surface was gradually submerged, and over it was accumulated stratified sand to an average depth of 600 feet.

(3). As the fauna of the Nanaimo series is characteristically marine, it is concluded that the Comox formation was formed during an overlap essentially estuarine in nature.

(4). Five stages of this overlap can now be recognized in part of the district, where they are represented by coal seams. Each coal seam is considered to represent an interval when the general subsidence was temporarily halted, allowing sedimentation to build up an even surface not far from sea level, after which the region remained stationary long enough for the coal seams to form.

(5). The original inequalities of the pre-Cretaceous were somewhat subdued by the transgressing waters, the tops of the hills being sliced off and the material being deposited in hollows, some of which were thus partially filled with coarse angular basal breccia or basal conglomerate.

(6). The coal seams range from a few inches to 25 feet in thickness, and individual seams vary from place to place in thickness and amount of coal. Some of the seams split and re-unite.

(7). Immediately following the accumulation of the Comox formation a more pronounced subsidence took place, marine conditions supervened, and the mud and silt which later were consolidated to form the shales of the Trent River formation were deposited over the coal measures.

(8). The present location of the workable areas of coal has been influenced by the original inequalities of the pre-Cretaceous surface to a greater extent than by the post-Cretaceous folding.

## THE NANAIMO SERIES.

The coal bearing rocks of southern Vancouver Island are of Upper Cretaceous age, and to them the name Nanaimo series (group) was given by Dawson in 1890[1]. Rocks of the Nanaimo series extend along the southeast coast of Vancouver Island and the adjacent islands from north of Campbell river to near Victoria, a distance, in an air line, of at least 150 miles. They are distributed in three areas or basins which are now fairly distinct, but which were formerly more co-extensive than is now the case. These basins are, from north to south, the Comox area[2], the Nanaimo area[3], and the Cowichan area[4]. The two northern basins contain

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([1]) Dawson, G. M., Am. Journ. Sci., Vol. 39, 1890, p. 181.
([2]) Richardson, James, Geol. Surv. Can., Rept. Prog., 1876-77, p. 161.
([3]) Idem., p. 170.
([4]) Clapp, C. H., Geol. Surv. Can., Memoir 96, 1917, p. 219.

*Table of Formations of the Nanaimo Series*[1]. *(Clapp)*

| Name | Lithological Character | Average thickness |
|---|---|---|
| Gabriola formation. | Chiefly thin to thick bedded yellowish-grey fine to medium sandstone. | 1,400 feet. |
| Northumberland formation. | Grey sandy shales at top and bottom, with a middle portion of sandstone and coarse conglomerates. | 1,150 " |
| De Courcy formation. | Chiefly greenish-grey, fine to coarse sandstone. | 900 " |
| Cedar District formation. | Chiefly grey, carbonaceous and ferruginous sandy shales. | 750 " |
| Protection formation. | Chiefly thick to thin bedded, greyish-white, fine to medium sandstone. | 650 " |
| Newcastle formation. (Douglas Coal Seam.) | Varies from a greenish grit to dark green sandy shales and shaly sandstones. Contains Douglas Coal Seam. | 175 " |
| Newcastle Coal Seam. Cranberry formation. | Chiefly thin bedded to shaly, dark green fine to coarse sandstones, with interbeds of sandy shale and fine agglomerates. | 400 " |
| Extension formation. | Chiefly quartzose conglomerate. | 600 " |
| Wellington Coal Seam. East Wellington formation. | Chiefly flaggy, olive-grey, fine to medium sandstone. | 35 " |
| Haslam formation. | Chiefly thin bedded. light to dark grey sandy and carbonaceous shale, with coarser rocks towards the base. | 600 " |
| Benson formation. | Basal conglomerate. | 100 " |
| | | 6,760 " |

(1) Idem., p. 45.

large fields of workable coal, but only a few thin unworkable seams are known in the Cowichan basin.

The type section for the Nanaimo series is the Nanaimo area, where it is stated by Clapp[1] to consist "of conglomerates, sandstones, and shales, which are almost invariably sandy, and some coal. In the Nanaimo map-area the series has a total average thickness of 6,760 feet and has a maximum thickness of 9,000 feet. In other places on Vancouver Island, and on the small islands off the east coast, the total thickness of the series is nearly 10,000 feet". Clapp's subdivisions of the Nanaimo series are given in the table on page 387.

In the Nanaimo area, "coal occurs chiefly in the lower part of the Nanaimo series, in three seams, the Wellington, the Newcastle, and the Douglas. The lowest seam, the Wellington, lies at the base of the Extension formation and is about 700 feet above the base of the Nanaimo series. The Newcastle seam occurs at the base of the Newcastle formation, and overlies the Wellington seam by about 800 to 1,000 feet. The Douglas seam is contained in the Newcastle formation, from 25 to 100 feet above the Newcastle seam"[2].

In the Comox area, to which the district described in this paper belongs, Richardson has divided the Nanaimo series into seven divisions or formations, as follows[3]:

G. Upper conglomerate.
F. Upper shales.
E. Middle conglomerate.
D. Middle shales.
C. Lower conglomerate.
B. Lower shales.
A. Productive measures.

In harmony with modern usage, as a result of his reconnaissance of 1911, Clapp re-named the rocks of divisions A and B the "Comox formation" and the "Trent River formation" respectively[4].

The detailed correlation of the formations of the Comox and Nanaimo areas must be left until the geology of the intervening region is studied more carefully than has yet been

(1) Clapp, C. H., Geol. Surv. Can., Memoir 51, 1914, p. 44.
(2) Clapp, C. H., Geol. Surv. Can., Mem. 51., 1914, p. 96.
(3) Richardson, J., Geol. Surv. Can., Rept. Prog., 1872-73, p. 35.
(4) Clapp, C. H., Geol. Surv. Can. Summ. Rept., 1911, p. 105.

done; but there seems no doubt that the Protection formation of the Nanaimo area and the Comox formation of the Comox area are identical, and only the sure knowledge of areal continuity is lacking to make the correlation certain, for they are entirely similar lithologically, and they are overlain by a similar succession of formations in both districts. In the Nanaimo area, however, the Protection formation is underlain by nearly 2,000 feet of Upper Cretaceous sediments, but in the Comox area the Comox formation is the basal division and lies directly on the pre-Cretaceous rocks. These relations indicate a pronounced overlap of the Nanaimo series from south to north, a theory which is substantiated by other evidence.

## CUMBERLAND AREA.

At the time of Richardson's examination, the most important geographic name of the region described in this paper was "Comox", and accordingly he used it to designate the area as a whole. Since then, however, the coal mining industry has centred, not at Comox, which is a farming village, but in and around the city of Cumberland. On this account, then, while the name Comox is retained as a general designation for the coal bearing area as a whole, and though it has been applied as a formation name to the productive coal measures, the district with which this paper is concerned is more accurately described as the Cumberland area.

*Pre-Upper Cretaceous Rocks.*—The basement on which the Upper Cretaceous sediments rest unconformably is formed of altered and metamorphosed volcanic rocks, consisting principally of agglomerates, tuffs, and flows, the latter frequently exhibiting the ellipsoidal structure known as pillow lava. These volcanic rocks, collectively known in the district as trap, are dark bluish-green to greenish-black; are very hard and tough under the hammer; and are resistant to the weather. They are probably of andesitic and basaltic composition. They outcrop extensively in the Beaufort range of mountains which lie west of, and steeply surmount the east coast lowland of, Vancouver Island in this region. In the Cumberland area all that is known of the age of these rocks is that they are pre-Upper Cretaceous, but on account of their lithology, degree of metamorphism, structure, and general appearance,

they are correlated with great confidence with the Vancouver volcanics of the Vancouver group, which is known to contain Lower Jurassic and probably Triassic rocks.

*The Comox Formation.*—The Comox formation underlies a strip of country from one and a half to two miles wide extending with a southeast trend continuously from Brown to Tsable river and beyond. Its southwestern boundary, where it is in contact with the volcanic rocks of the Vancouver group, is irregular; embayments of the Cretaceous extend back into the pre-Cretaceous rocks, and outliers of Comox sandstone are found lying on and surrounded by the older metamorphic volcanics. The northeastern boundary, where the Comox formation is conformably overlain by the Trent River formation, is more regular.

The Comox formation is dominantly composed of white sandstone, varying from a chalky white to pale grey, and from fine to medium in grain, though coarse sandstones and grits are occasionally found. Grey and brownish-grey clay shales and carbonaceous shales occur, typically associated with the coal seams. In several places a basal breccia of heterogeneous texture, formed from angular fragments of the underlying volcanic rocks, has been noted, but only in one instance, on Trent river, has a basal conglomerate with rounded pebbles been observed. In the vicinity of Tsable river a band of intraformational conglomerate is found in the sandstone. This conglomerate is composed of well rounded pebbles of argillites, quartzite, quartz, and various granitic rocks, from a quarter of an inch to two feet in diameter, but usually under two inches, and embedded in a sandy matrix. It will be further mentioned below. The sandstone of which the greater bulk of the formation is composed is remarkable for the thickness and homogeneity of its beds, some of them having been observed up to 40 feet thick without a visible stratification plane, and beds 10 to 20 feet thick are not unusual. Where such a sandstone lies above a workable seam of coal, it forms an excellent roof.

Between the Comox formation and the underlying volcanic rocks is a marked unconformity. The contact of the two formations has been observed in many places in the streams, and its nature is known in considerable detail in the vicinity

of Tsable river, over an area of about 20 square miles, both from surface exposures and borehole records. The information gained from this area proves what had previously been inferred, that the surface on which the Upper Cretaceous rocks were accumulated was one of considerable relief, amounting to 440 feet. The consequences of this are material; for not only does this cause the Comox formation to be of irregularly variable thickness, but on account of the fact that the best coal seams occur in the lower part of the formation, it means that these seams are absent in areas where considerable elevations of the pre-Cretaceous surface existed, because these areas stood up above the even floor caused by the spread-out sediments, which was a necessary factor in the accumulation of the seams. The general relations of the Comox formation to the pre-Cretaceous rocks is well illustrated by Figure 1, which

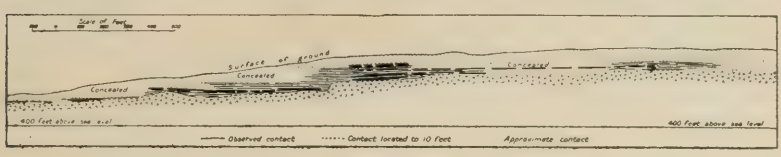


Figure 1

represents a well-exposed vertical section along Coal creek, a small stream near Cumberland. This section cuts across the irregular surface of a flank of a pre-Cretaceous hill, which was

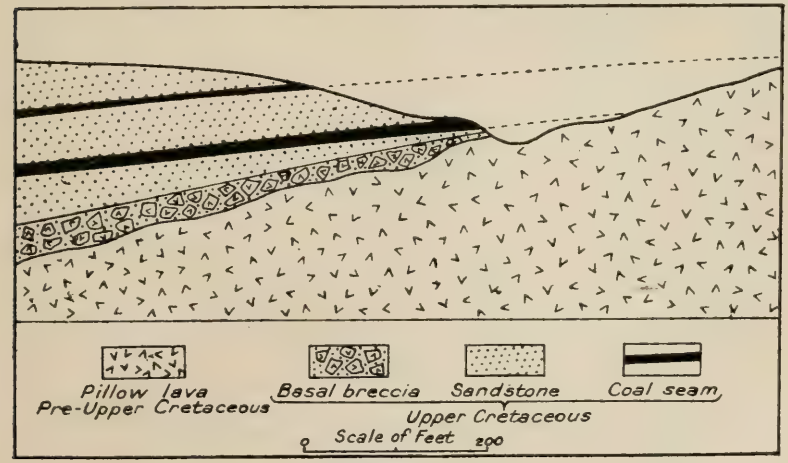


Figure 2

gradually overlapped and buried by the encroaching flood of sediment as the old land sank. In several instances depressions in the pre-Cretaceous floor have been filled with breccia of the volcanic rocks, as illustrated in Figure 2. In this occurrence, observed on Tsable river, an old knob of pillow lava is overlain at one place by about five feet of fine pearl-grey shale, capped by a coal seam seven feet thick. Five hundred feet distant there is at least 100 feet of strata between this coal and the pre-Cretaceous rocks. These intervening strata consist of a band of sandstone underlain by a coarse basal breccia, both of which bands wedge out as they approach the original knob. This filling up of the more pronounced minor depressions in the old surface is a characteristic feature of the basal Upper Cretaceous unconformity, and has been previously noted by Richardson[1], Dawson[2], and Clapp[3].

In several localities, and notably on Coal creek as illustrated by Figure 3, the surface of the pre-Cretaceous volcanic rocks is decomposed to a depth of ten feet or so. Although the older volcanics are ordinarily hard and resistant imme-

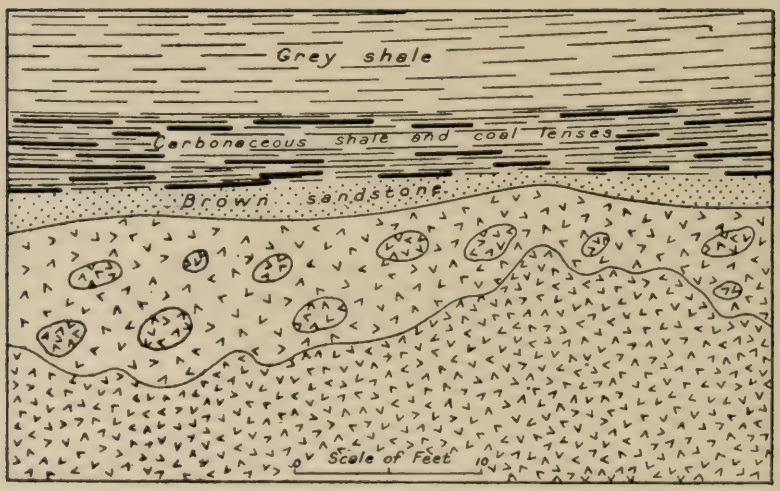


**Figure 3**

(1) Richardson, J., Geol. Surv. Can., Rept. Prog., 1876-77, p. 167.
(2) Dawson, G. M., Geol. Surv. Can., Ann. Rept., Pt. B. 1886, p. 42B.
(3) Clapp, C. H., Geol. Surv. Can., Mem. 51, 1914, p. 76.

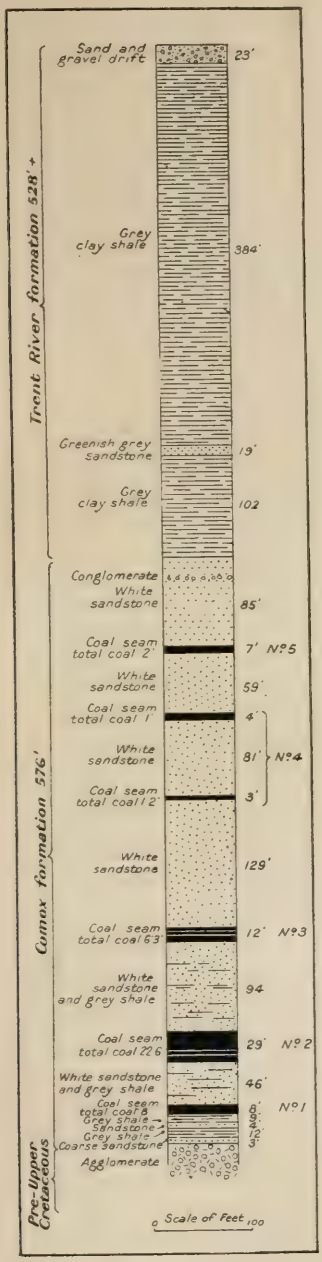
Sand and gravel drift
23'
Grey clay shale
384'
Greenish grey sandstone
19'
Grey clay shale
102
Trent River formation 528'+
Conglomerate
White sandstone
85'
Coal seam total coal 2'
7'
No 5
White sandstone
59'
Coal seam total coal 1'
4'
White sandstone
81'
No 4
Coal seam total coal 1 2'
3'
White sandstone
129'
Coal seam total coal 6 3'
12'
No 3
White sandstone and grey shale
94
Coal seam total coal 2 2 6
29'
No 2
White sandstone and grey shale
46'
Coal seam total coal 1 8
8'
Grey shale
4'
Sandstone
12'
Grey shale
Coarse sandstone
3'
No 1
Comox formation 576'
Agglomerate
Pre-Upper Cretaceous
0 Scale of Feet 100

Figure 4

diately below the Cretaceous rocks, at these places there intervenes a zone of soft clayey material, holding rounded nodules of undecomposed volcanic rock, which are surrounded by concentric shells of partially decomposed rock, the whole lying with a sharp contact on the fresh-looking volcanics below. The conditions described strongly suggest that the pre-Cretaceous surface was weathered to a depth of several feet at least, and at the localities mentioned this weathered surface has been preserved by burial beneath the Cretaceous sediments. A characteristic feature of the basal Comox formation, illustrated by Figures 1, 2, and 3, is that apart from the basal breccias and conglomerate, which are exceptional and local, the lowest strata are typically fine grained: sandstone, shale, and even shaly coal, having been seen lying directly on the pre-Cretaceous surface.

A columnar section of the Comox formation is given in Figure 4, which has been only slightly generalized from borehole records. It illustrates graphically the great predominance of sandstone, and also the general vertical distribution of the coal seams in the Tsable River field.

The intraformational conglomerate, the usual stratigraphic position of which is near the top of the formation, varies from a feather edge to 600 feet thick, but is usually a sheet of some 60 feet in thickness. Although it is conformable on the whole, it locally fills hollows cut by contemporaneous erosion, and in one part of the area it has cut out nearly the whole thickness of the Comox formation, including, of course, the contained coal seams. Figure 5 illustrates the mode of

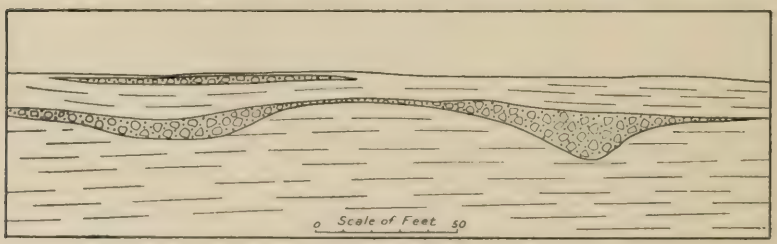


**Figure 5**

occurrence of this conglomerate where exposed at the falls on Tsable river. At this place the conglomerate plainly fills an

old stream channel cut in the previously deposited sandstone. Contemporaneous erosion in the Comox formation has also been observed in another locality, as shown in Figure 6.

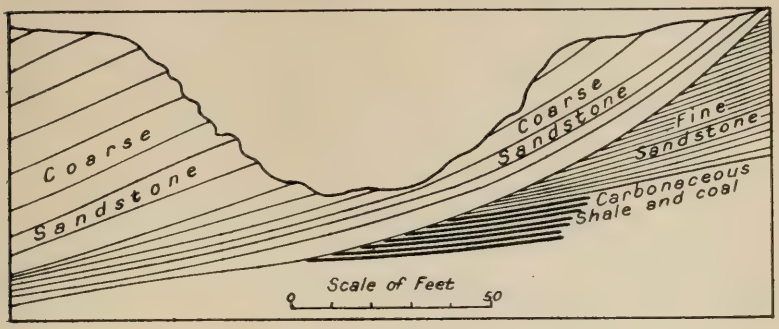


Figure 6

Here, the coal seam, at this point poor and shaly, has been in part removed by a current carrying coarse sand which has been deposited, with pronounced false bedding, in the stream channel. Minor false bedding is a characteristic feature of almost every layer of sandstone in the Comox formation, and can be readily detected on the slightly weathered surfaces of the stream canons.

As stated, the Comox formation was accumulated on an irregular surface which caused a variation in its thickness. This is a notable feature of the formation wherever an area underlain by its complete thickness has been explored by the drill, and a typical variation is illustrated in Figure 11. Thicknesses ranging from 80 to 1,000 feet have been recorded in the drill holes, though a fair average for the formation is about 600 feet.

Richardson did not find any fossils in his Productive Measures[1], and they are undoubtedly rare, though some pelecypods occur in the roof of a seam mined at Cumberland. These have not yet been specifically determined.

*The Trent River Formation.*—The territory occupied by the Trent River formation is a strip along the coast with a fairly constant width of four miles, its southwest boundary being the top of the Comox formation.

(1) Richardson, J., Geol. Surv. Can., Rept. Prog., 1872-13, p. 44.

In striking contrast to the underlying sandy coal measures, the Trent River formation is dominantly (and in certain places wholly) composed of shale, with occasional fine grained sandstone bands. Between Trent river and Cumberland, and again in the area between the town of Union Bay and Langley lake, there are intercalations of conglomerate and sandstone several hundred feet thick. A characteristic feature of the shales is the abundance of sandstone dykes in the lower portion of the formation. Some of these are continuous for scores of yards, though often only a few inches and in some instances only a fraction of an inch thick. With the interbeds of sandstone, always present in the zones where the dykes occur, they form a network of hard sandy ribs in the shale. They are usually, though not always, perpendicular to the stratification.

The shales and sandstones are finely and evenly laminated. Fossils — ammonites, pelecypods, and belemnites — are found in numerous localities and though the character of the shale matrix is such as to make their collection difficult, enough have been obtained by Richardson[1] to determine the marine origin of the formation.

The summit of the formation is not exposed in the area studied, but outcrops on Denman island, which lies across Baynes sound to the northeast. Richardson gives 1,000 feet as the thickness[2].

The divisions of the Nanaimo series above the Trent River formation are exposed on Denman and Hornby islands and are described by Richardson[3].

So far as has been observed, the Trent River formation is conformable on the underlying Comox sandstones. Local unconformities have been noted in the formation itself, however, in connection with the conglomerate and sandstone areas mentioned above.

*Post-Upper Cretaceous Intrusive Rocks.*—In the northwest corner of the area examined, an eminence called Anderson hill is formed of a mass of igneous rock which has broken through the Comox formation. It is a grey, speckled porphyry, with distinct euhedral phenocrysts of plagioclase, up to one-eighth

(1) Richardson, J., Geol. Surv. Can., Rept. Prog., 1872-73, pp. 45-47.
(2) Richardson, J. Geol., Surv. Can. Rept. Prog., 1876-77. p. 162.
(3) Richardson, J., Geol. Surv. Can., Rept. Prog., 1872-73, pp. 47-51.

of an inch across, in a dense matrix, and it has been called by Clapp[1] a dacite porphyry. On the precipitous north slope of Anderson hill, the igneous mass clearly consists of at least two sills, the lower being 30 feet thick, and separated from the upper one by 40 feet of sandstone. The upper sill, which may consist of more than one intrusion, is at least 200 feet thick. Boreholes on the south end of Anderson hill strongly suggest crosscutting intrusive relations, so this igneous body, though in part sill-like, may be nevertheless of an irregular shape. In the abandoned mine at Bevan, where the old workings penetrated to near the south end of Anderson hill, the coal was found to have been affected by this intrusion, and though some of it was rendered valueless, much of it was improved in quality to a semi-bituminous nature. On Brown river, a sill of similar porphyry, 3 to 5 feet thick, has been injected into a coal seam, which originally was about 5 feet thick. The effect of this intrusion changed the seam to a mass of impure carbonaceous and sandy material, quite valueless as coal.

*Structural Geology.*—The structure of the Nanaimo series in this area is in general a monocline of gently northeastward dipping strata, modified in some places by folding, and also by faults of small displacement. The average dip is about five degrees, but, locally, northeastward inclinations up to 50 degrees have been observed, and in some parts of the field low dips in the opposite direction occur. In the vinicity of Cumberland the general northeast slope of the trap floor is 2 to 3 degrees, and in the Tsable River section it is 6 to 7 degrees. This section is affected by undulatory folds, and at least two synclinal and one anticlinal axes cross the field in a direction not quite parallel to the general strike. The distribution of the coal-bearing areas, however, seems to be only secondarily affected by the folding, and apparently the original inequalities of the pre-Cretaceous surface are of greater significance with respect to areas of workable coal than is the post-Cretaceous folding. The folding in the Cumberland area is not so pronounced as in the Nanaimo field.

No large faults have been detected, but several small normal faults with a displacement of less than a score of feet

(1) Clapp, C. H., Geol. Surv. Can., Sum. Rept., 1911, p. 106.

have been noticed in the field. In one of the Cumberland mines a number of normal faults occur in such a way as to suggest that they lie in a zone that has been subjected to tensional stress. These faults range from a few inches to 20 feet in displacement, and are of textbook sharpness, forming breaks of great precision. They occur in a seam about 100 feet above the base of the measures, and it is not known whether they extend through to the lowest seams. It is thought that they do not so extend, for in closely adjacent areas where the lowest seam (which is only a few inches from the trap) is mined, no faults are found.

It is probable that these faults are caused by localized anticlinal bulging of the basement rocks, causing the overlying layers of sediments to be stretched and cracked, and the arches so formed to collapse. The packing and compression of the measures from settling caused by the expulsion of the water originally laid down with the sediments, may have been a contributory factor, although, because of the relatively small thickness of mud sediment (in which such settling is much more pronounced than in sands), such a cause will hardly serve wholly to explain the displacements observed.

The rocks are intersected by comparatively few joints, noticeably less so than in the Nanaimo area, and this is especially the case in the Comox sandstones.

*The Coal Seams.*—During the stage of the accumulation of the Comox formation there were recurrent episodes when coal seams formed. In the Tsable River area, five of these episodes may be clearly recognized by five usually quite distinct seams which can be correlated with some degree of assurance. In the vicinity of Cumberland the coal accumulation is not so clearly recognizable as belonging to distinct intervals, but even here there is some evidence that the thicker coal seams at any rate were accumulated at four or five separate horizons. It is also generally, though not always, true that the thicker seams were formed near the base of the measures, and that the seams formed later are thin and unworkable.

Characteristically, the coal is associated with layers of grey or brownish-grey shale. Rarely, a band of clean coal is enclosed between a sandstone roof and floor, there being nearly always more or less shale above or below the seam, and fre-

quently the coal is wholly enclosed in shale. Like the seams in other parts of Vancouver Island, these have no trace of anything resembling underclays, nor have rootlets, tree stems, branches, or leaves been observed in association with the coal. In fact the only plant fossils seen, other than thecoalitself, were not in place, one being a nodule containing poorly preserved fern impressions, and the other a piece of a tree stem surrounded by an outer layer of bright coal as if the bark had been coalified. Apart from the clay shale associated with the seams, more or less fissile carbonaceous shale, and the brown compact shale known as 'bone', occur interbedded with the coal itself. These impurities vary from a lamina, of paper thinness, to bands occupying most of the thickness of the seam; and instances occur where the seam consists of shale, or of coal so high in sediment as to be unworkable.

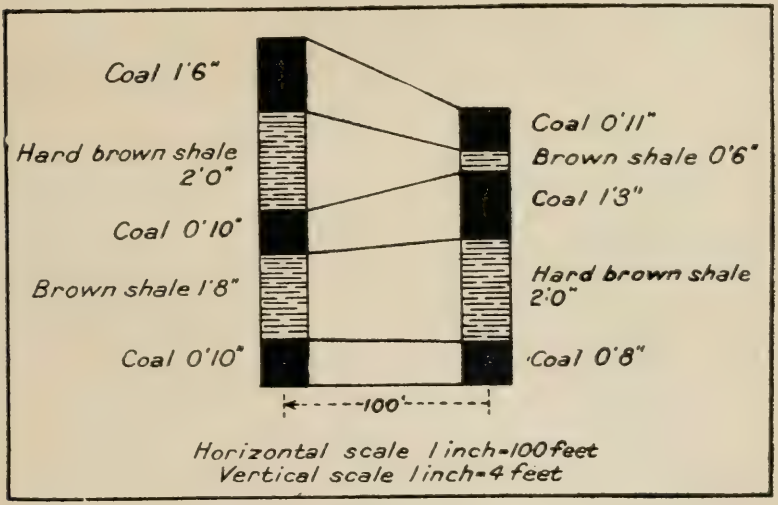


Figure 7

This is particularly the case where the seam closely approaches the pre-Cretaceous rocks. Neither in the outcrops nor in the boreholes has a clean seam of coal been observed resting directly on the old volcanics, though dirty coal, or shale with coaly streaks, frequently does so. It is obvious that near the landward boundary of the areas in which the seams formed they are mixed with rock debris to such an extent that they are valueless as coal. In one of the mines at Cumberland

working in a seam usually 100 feet above the base of the measures, the coal becomes thin, poor, and unworkable when it overlies one of the protuberant knobs of the old floor, and further progress in this area has been abandoned.

Lateral variation in the seams is illustrated in Figures 7 and 8.

Figure 7 is a diagram of a dirty seam, the lowest exposed on the Trent river; and Figure 8 illustrates a seam exposed on Tsable river. The coal at this place on Tsable river is bright, clean, hard, and evenly laminated to a degree unusual on Vancouver Island. Both of these seams are within a few feet of the pre-Cretaceous basement rocks where sectioned, the seam of Figure 8 being the lower one shown in Figure 2. Where workable coal passes into shale or unworkable coal in the mines, the areas thus barren are termed 'wants'. The distribution and other relations of these wants, subjects of much importance to the coal miner, have not yet been studied.

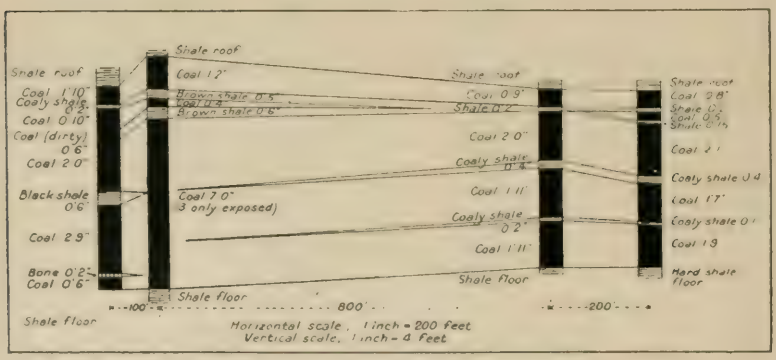


**Figure 8**

Still another characteristic of some of the seams is the manner in which they split and reunite through the intercalation of lenticular layers of shale or sandstone, a feature that is apparent from a study of the borehole records, some of which are indicated in Figure 11. The thickness of coal in any given steam may vary from a fraction of an inch to many feet, 25 feet of coal being the thickest obtained in any single seam. This, however, included a band of shale four inches thick, and the coal was soft and shaly. A solid bench of bright hard clean coal exceeding 30 inches in thickness is an unusual occurrence,

though benches up to 35 inches are known. The upper of the two seams now being mined at Cumberland averages 3 feet 8 inches of coal; the lower one, about 100 feet below and near the base of the measures, averages 4 feet 2 inches.

The interesting question of whether transportation or growth in place better explains the accumulation of these seams will be left for discussion at a future time.

*The Conditions of Coal Accumulation.*—The manner in which the Comox formation was accumulated over an irregular surface in successive stages marked by coal seams is well illustrated in that part of the area known as the Tsable River coalfield. This district is for the most part thickly covered by till or stratified sand and gravel drift, and the only natural exposures of bedrock are in the streams and on some of the steepest slopes. A large number of diamond drill holes, however, have afforded much detailed information about the coalfield, and in nearly every case they penetrate through the coal measures into the underlying pre-Cretaceous rocks. These excellent 2-inch cores furnish complete and continuous sections of the formations passed through by the drill, and every seam of coal down to a fraction of an inch is thus detected. These boreholes supplement the natural exposures, and the information gained from both leads to interesting and significant conclusions.

The areal bedrock geology of a part of the Tsable River field is shown in Figure 9, which also shows the outcrop of the coal seams. Where observed in the field, these are indicated by solid lines, and by dashed lines where their location has been deduced from borehole records. It is noticeable that the basal contact of the Comox formation is irregular. This is primarily due to the original inequalities of the old pre-Comox surface, though it is secondarily modified by folding. The amount of the irregularity due to folding is indicated by the curvature of the boundary line between the Comox and the Trent River formations, a line that may be supposed to have been originally straight. It is also apparent that the outcrop of the highest coal seam closely parallels this contact, and that the lower seams have a more irregular trace.

Figure 10, which is to be compared with Figure 9, is a contoured map showing the present topography of the surface

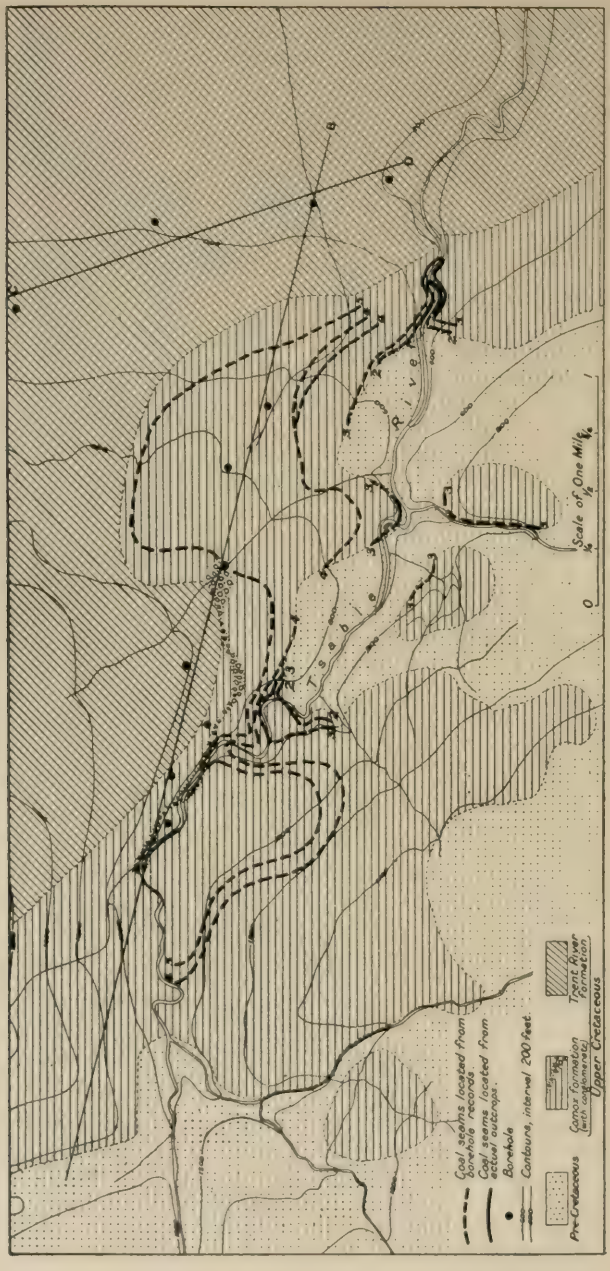
Coal seams located from borehole records
Coal seams located from surface outcrops.
Borehole
Contours, interval 200 Feet.
Pre-Cretaceous
Comox Formation (with conglomerate)
Trent River Formation
Upper Cretaceous
Scale of One Mile
0

Figure 9

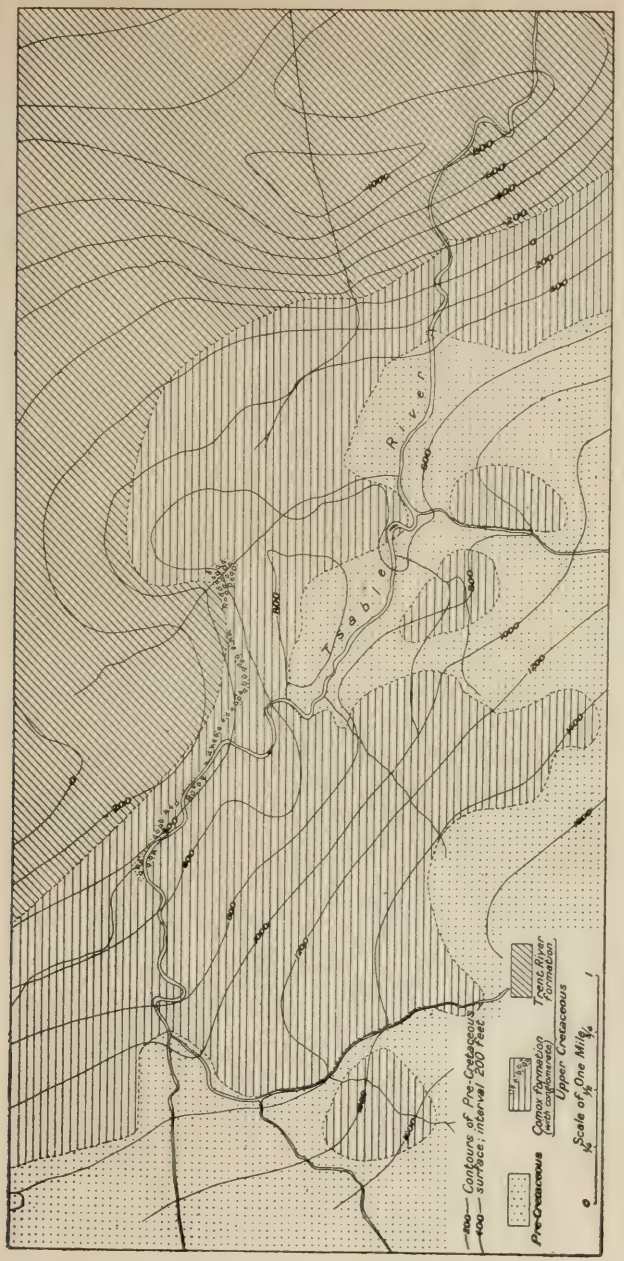
Tsable River
Contours of Pre-Cretaceous surface; interval 200 Feet.
Pre-Cretaceous
Comox Formation (with outcrops)
Upper Cretaceous
Trent River Formation
Scale of One Mile

Figure 10

of the pre-Cretaceous rocks. This map, in the areas covered by the Cretaceous formations, gives the underground structure, and over the areas occupied by pre-Cretaceous rocks it is a restored surface. The underground structure map is adapted from one made by Mr. A. C. Lynn of the Canadian Collieries (Dunsmuir) Ltd. It was made by referring the position of the trap surface, as obtained from borehole data, to sea level as a datum, and plotting the contours with an equal vertical interval of 100 feet. The restored surface represents what, in my own judgment, would be the present position of the pre-Cretaceous surface had it undergone no erosion. Wherever the present-day surface contours shown in Figure 9 cross the basal contact of the Comox formation, they give the present position of the pre-Cretaceous surface. Inside of the areas underlain by the Cretaceous rocks, the pre-Cretaceous surface contours must of necessity be further 'upstream' than are the present-day surface contours, because the pre-Cretaceous surface is buried; outside of these areas, the pre-Cretaceous surface contours must be farther downstream than are the present-day contours, because the pre-Cretaceous surface is eroded. Pre-Cretaceous and present-day contours cross, therefore, at the contact of the base of the Cretaceous on the underlying rocks. Inside the Cretaceous areas they have been located either from borehole data or from inferences which need not be detailed here; and outside these areas smooth flowing, reasonably spaced lines have been drawn across the eroded pre-Cretaceous from one point on the contact of one Cretaceous area to the corresponding point on the contact of the next adjacent, and so on.

Figure 10 represents, as stated, the present topography of the pre-Cretaceous surface, but it differs from the pre-Cretaceous topography because that surface has been tilted since, and moderately folded. As the problem of how to correlate the coal seams depends on the determination of the extent to which the present irregularity of the pre-Cretaceous surface is due to folding or to the original inequalities, it is important to arrive at a correct solution. In the areas now covered by the Trent River formation which have been explored by the drill, the following method has been used to solve this problem.

Two vertical structure sections on the lines A,B, and C,D, of Figure 9 are given in Figure 11. The section on C,D,

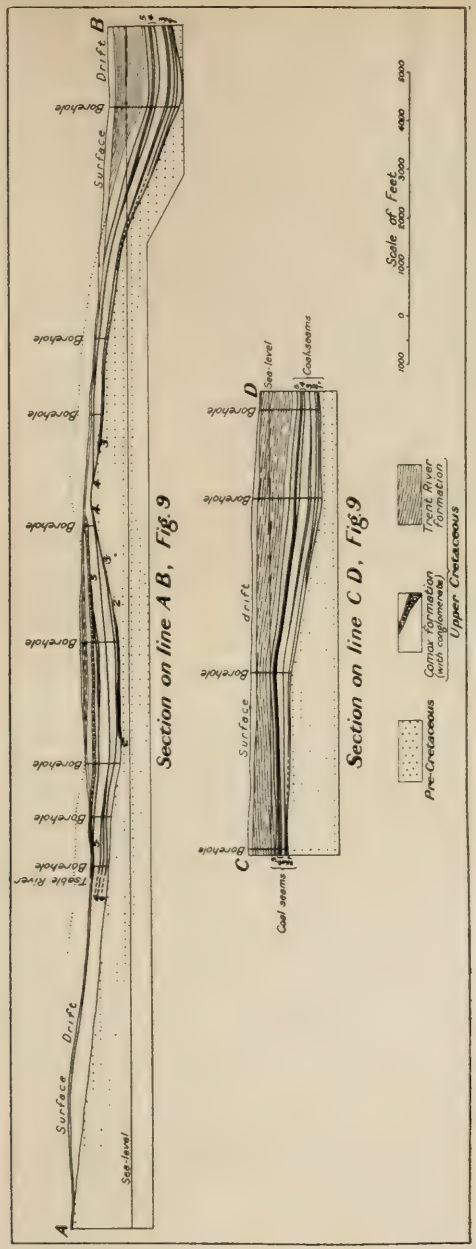
Section on line A B, Fig. 9
Section on line C D, Fig. 9
Drift
Surface
Sea-level
Borehole
Tsable River
Coal seams
Sea level
Scale of Feet
1000 0 1000 2000 3000 4000 5000
Pre-Cretaceous
Comox Formation (with conglomerate)
Trent River Formation
Upper Cretaceous

Figure 11

exemplifies the variable thickness of the Comox formation, which variability has been already explained as being caused by the original inequalities of the surface on which it was laid down. The Comox formation was accumulated to a thickness more than sufficient to bury the inequalities of the old terrane in the area it now occupies, so that its upper surface may be supposed to have been an even one, and nearly level. The supposition is greatly strengthened when it is considered that the lower Trent River formation is composed of finely laminated shales and fine continuous interbeds of sandstone, which were undoubtedly deposited in a nearly flat condition. The line of contact between the Trent River and the Comox formations may thus be taken as having been originally straight, and virtually horizontal. If this line, therefore, be plotted horizontally as shown in Figure 12, and the thickness of the Comox

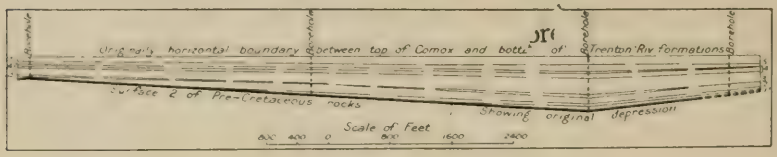
Originally horizontal boundary between top of Comox and bottom of Trenton Riv formations
Surface of Pre-Cretaceous rocks showing original depression
Scale of Feet
800 400 0 800 1600 2400

**Figure 12**

formation, as determined from boreholes, be plotted below it, the extent of the original inequalities in the surface is apparent. If this truly represents the conditions when the Comox formation was being accumulated, the coal seams should appear as horizontal parallel lines, and it will be observed that the upper seams are in fact thus related. The lower seams are not parallel, however, but are observed to diverge toward the thicker portions of the formation accumulated over the deeper hollows in the old pre-Cretaceous floor. This lack of parallelism is the result of the operation of two factors in sedimentation, and although their relative influence is rather difficult of quantitative determination, they are thought to be sufficient to cause the observed divergence of the seams. In the first place, if an uneven surface be subjected to sedimentation, the sediment will form a series of strata at first closely paralleling that surface, but later, by slipping on the slopes, or by currents being deflected away from the eminences, or by undertow, or by wave action, the layers tend to thicken over the deeper

portions of the area. The second factor in causing the strata to sag over the hollows is the settling of the sediments brought about by the weight of the superincumbent measures expelling their original water content. This of course takes place throughout the prism of sediments, but is more marked where the prism is thicker; there the differential effect causes the sag in the lower measures, the upper seams tending more nearly to approach a straight line in section. A sag of this nature has been repeatedly observed in sections constructed in this manner in the Cumberland, as well as in the Tsable River, areas.

Wherever the Comox formation is covered by the Trent River formation, so that the full thickness of the coal measures is assured, this method enables the coal seams to be correlated with some confidence. In that portion of the Tsable River coal field underlain only by the Comox formation, the degree to which it has been folded is not susceptible of direct determination, since the natural exposures are not sufficiently abundant to permit use of observations of strike and dip. For the correlation of the seams, therefore, other principles, such as the general similarity of lithology, thickness, and stratigraphic succession of measures between successive seams, must be applied. The complete absence of diagnostic fossils, either animals or plants, adds to difficulties of correlation, as does the fact that the seams themselves vary in thickness from place to place even over short distances. It has been noted, however, that though the thickness changes, the seams are laterally widespread, and a coal seam established in one borehole is almost certain to be found in neighbouring ones, even if only a few inches thick.

After assembling and considering all the available data, it is apparent that the coal formed along the edge of an Upper Cretaceous land having an irregular relief up to 440 feet. Owing to its subsidence, this land was gradually covered more and more by the encroaching sediments, and Figure 13 shows the areas over which the coal seams were successively accumulated. It affords also, in effect, a contour map of the pre-Cretaceous surface, since any single seam may be considered to have been accumulated in a virtually horizontal area. The contour interval of the map is different and variable for

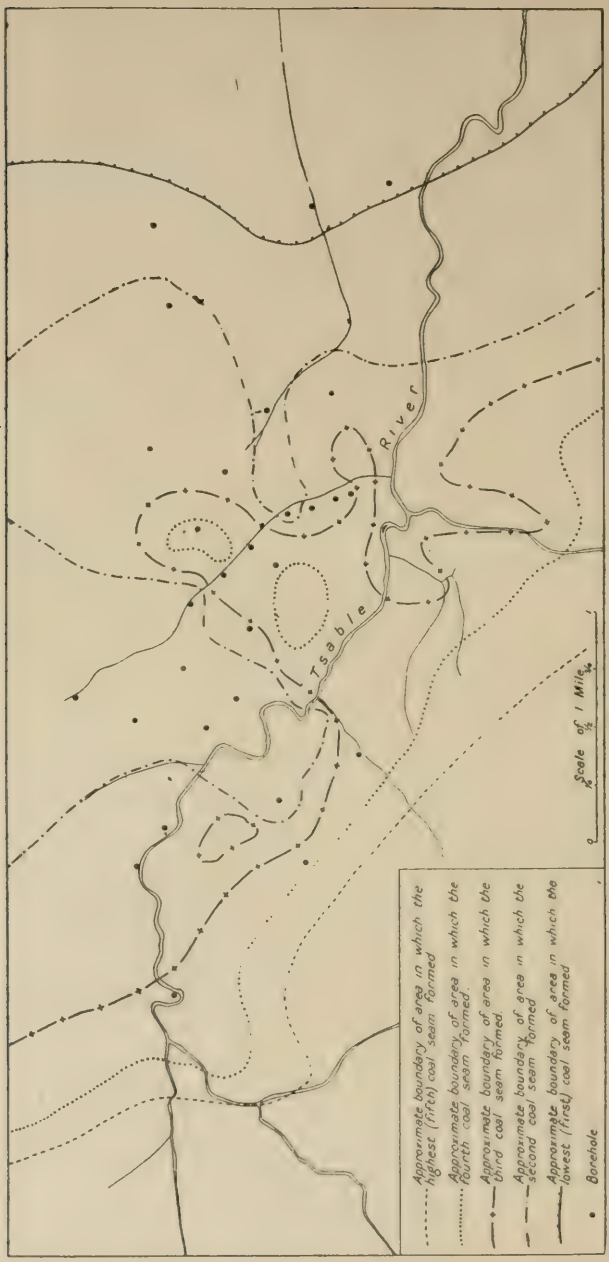
Isable River
Approximate boundary of area in which the highest (fifth) coal seam formed
Approximate boundary of area in which the fourth coal seam formed
Approximate boundary of area in which the third coal seam formed
Approximate boundary of area in which the second coal seam formed
Approximate boundary of area in which the lowest (first) coal seam formed
Borehole
Scale of 1 Mile

Figure 13

each pair of seams; from the first to the second seam it averages 60 feet; from the second to the third, the average is 100 feet; from the third to the fourth, 150 feet; and from the fourth to the fifth, 130 feet. This gives a total known relief in this small area of 440 feet, without allowing for the slight depth of the basin below the lowest seam, or for the height of the land above the highest.

So far as is now known, the general condition over the distance of 24 miles from Brown river to Cook creek were similar during the accumulation of the Comox formation, so the conditions under which the Tsable River field was formed may be considered as typical of the whole region. It is thus apparent that the pre-Cretaceous land sank, and over it were accumulated the sands and muds and vegetable matter which now form the Comox formation. This information is the result of an overlap, which continued at least throughout the time that it took to accumulate the sediments now represented by an average thickness of 600 feet, and which halted at five successive intervals during which coal seams formed.

It is hoped that further field work will afford the information that will enable to extent and significance of this overlap to be discussed more fully.

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## DISCUSSION.

THE CHAIRMAN (MR. THOMAS GRAHAM): A year ago, in an address I had occasion to give before the members of the Vancouver Board of Trade, I referred to some of the difficulties we are meeting with in this field. Some people thought I was simply attempting to explain a boost in the price of coal, or trying to throw a smoke screen over our operations on Vancouver Island. I am very pleased to-day to see that Mr. MacKenzie, after a scientific investigation, practically confirms the statements made by me at that particular time. In the Cumberland field, the Dunsmuirs had 150 to 175 drill holes, so close together in places that they were almost like the holes in a sieve. Yet, with all these drill holes, they were not able to correlate the several seams. In many places a trap would be encountered at a depth of about 100 feet, with no overlying coal, and then, only 500 feet or so away, they would drill a

thousand feet without striking the trap, but passing through several seams of coal. They had three workable seams, one of which attained a thickness of 25 feet, and a few hundred feet distant from these there would be only 3 inches of coal. The difficulty of the problem of correlating these seams has been clearly demonstrated by Mr. MacKenzie. We hope in the future to have some measure of success in this direction, but even then the operation of the Cumberland field will always be attended with great difficulties. In the eastern coal fields, they have miles and miles of territory in which the seams are continuous and very little disturbed, whereas in the Cumberland area the attitude and dimensions of the seams have been determined by the condition of the original valley floor. Speaking of the folding in the Cumberland field, it has not been very pronounced. In many places we have a series of block faults, which are often spaced 7 or 8 feet apart. This is especially the case in No. 5 mine.

In conclusion, Mr. Graham referred to the difficulties of endeavouring to operate by longwall mining under these conditions, and called on Prof. Daniels, who had been three months in the field, to discuss the subject.

PROF. JOSEPH DANIELS: I spent two months in the Cumberland field making an examination of part of the area described in this paper. The principal difficulty, as already pointed out by Mr. MacKenzie, is in the correlation of the seams encountered in the boreholes scattered over a relatively large territory. The recognition that the line between the Trent and the underlying formation is the key to the interpretation of the correlation is a most important contribution by Mr. MacKenzie.

The topographic undulations of the underlying volcanics on which the sedimentaries were deposited are clearly recognized from field observation and from borehole evidence. The fact that the seams vary in thickness from a few inches to a few feet within short distances, together with variation in stratigraphic intervals, makes it extremely difficult to correlate in the absence of a marked or defined horizon; and until a base, such as the overlying Trent contact, was revealed, the oppor-

tunities for error were very great. In that portion of the Tsable River field where boreholes did not penetrate the overlying shales, correlation was subject to considerable chance of error.

The experience in this particular case emphasizes the importance of being very careful not to draw hasty conclusions regarding the commercial possibilities of coal-bearing areas. An extensive programme of prospecting, both by surface work and diamond drilling, may outline a potential coal-field; but it is not safe to assume that this has definitely proved that the area is capable of developing profitable mines.

# COAL MINING IN COMOX DISTRICT, VANCOUVER ISLAND.

By Charles Graham.

Annual General Meeting, British Columbian Division, Vancouver, February, 1922.

Coal mining on Vancouver Island has been actively carried on for over seventy years, principally at Nanaimo. Wellington, Northfield, South Wellington, Extension, and Cumberland. In the Nanaimo field there are three seams of coal, known as the Wellington, Newcastle, and Douglas seams, and one or more of these seams has been worked in each of the above mentioned districts, with the exception of Cumberland.

The present paper deals with the Cumberland field, or, as it is probably better known, the Comox field. This field occupies a strip three miles wide and six miles long, running from the Puntledge river in a southeasterly direction to just south of Cumberland, and situated along the western edge of the Cretaceous lowland which forms the east coast of Vancouver Island. In these 18 square miles there have been opened up eight mines, two of which are still operating.

## General Geology.

The underlying crystalline or trap rock of the area has been correlated with the Vancouver volcanics and determined as Jurassic in age. The relief of its surface just prior to sedimentation was rather pronounced. Some slopes as steep as 30 degrees existed, and there was a total relief of over 300 feet. Consequently, as the sediments were laid down on this uneven floor, the first beds had very irregular outlines, while those deposited later, after the depressions had been filled up, were more persistent areally.

These sedimentary rocks are of Cretaceous age and are divided lithologically into two formations, the *Comox* and the *Trent River*. The Comox formation consists essentially of sandstone, principally white in colour and with a calcareous

cement. It is in this formation that the coal seams occur. In all, there are more than ten seams, and they vary in thickness from 2 inches to 8 feet; but only three of them are over 3 feet and workable. These embrace the lower seam, now being worked in No. 4 mine; the upper bench of this seam, which has been worked in No. 4 mine; and what is known as the upper seam, which is the one No. 5 mine workings are in.

The Comox formation attains a thickness of over 700 feet in this area. Overlying it is the Trent River formation, consisting essentially of shale with thin sandstone beds distributed through it. This formation is of no importance as far as coal is concerned, as no workable beds have been found in it.

The whole lowland has been uplifted and given a northeast tilt. The basement rocks have also been folded, and the overlying sediments have conformed to these gentle open folds of the underlying rocks. The faults due to the folding are in most cases clean, sharp, normal faults, and they occur in zones, the most noted of which is the west slope of No. 5 mine. Here there is a regular network of faults having two general directions, N.60°-70°E. and N.10°W. Most of the faults have a vertical displacement of less than 10 feet, but in some the throw is much greater than this. These fault zones are the source of a great deal of trouble in working or developing the mine.

The measures above No. 4 mine dip to the northeast, while at No. 5 mine the dip swings round slightly to the north. This gives rise to a pitching syncline, and it is in this basin that the best coal is obtained. Over the anticline to the east the coal is very poor.

During the glacial period the area was covered with ice, which moved in an easterly direction to the strait of Georgia. The remnants of the ice may still be seen in the hills at the head of Comox lake. The glacier scoured the surface and then, in retreating, deposited a thin mantle of glacial till or boulder clay on the rocks. This in turn was covered with a coat of stratified drift, which is over 100 feet thick in places. A general uplift of the region followed, and subsequent erosion has given rise to the present-day surface.

## MINE OPERATION.

Of the eight mines which have been opened in this field, only two, Nos. 4 and 5, are now in operation.

*No. 4 Mine.*

With the exception of No. 1 mine at Nanaimo, this mine is the oldest and most extensive in the province. It covers 1,500 acres, and has been in operation since 1890. The workings are in what is known as the Lower seam, which outcrops on the shores of Comox lake and dips to the northeast, the average pitch of the seam being 7 degrees.

*No. 1 Slope.* — The mine is opened by two slopes. No. 1 slope is driven due north, half across the pitch, while No. 2 is turned off, just inside the mouth of the opening, at an angle of 45 degrees to the east, which is the full pitch of the seam. At a point 4,800 feet from the surface, No. 1 slope is turned parallel with No. 2, and driven in this direction to a point 9,400 feet from the surface. Levels have been turned off to the right and left, the lowest being Nos. 20 East and West levels. The slope has been driven about 200 feet beyond No. 20 West level.

At a point 6,000 feet from the surface, an upthrow fault was encountered. A level landing was made at this point, and a level rock tunnel driven through the fault. After coal was reached the slope was continued. The only part of the mine now operating is on the lower side of this fault. Nos. 16, 17, 18, 19 and 20 West levels are being worked, but all the workings above and on the east side have been either finished or abandoned. In order to consolidate the workings on the west side, and to improve haulage conditions, No. 15 West level (which was turned off just inside the fault and which had been abandoned) was reopened, and from it a slope, No. 4, was driven due north to cut off the faces of the levels below.

Until this fault was encountered, all the operations had been on the pillar and stall system; but as, on the inside of the fault, the coal was much thinner than on the outside, it was decided to change the method of working. Accordingly, when No. 4 slope was driven, cutting off the levels below No. 15, the system was changed over to longwall, and all subsequent operations conducted from this slope have followed this system.

At the present time two new slopes are being started below No. 20 West level. One of these is being driven N.45°E, and is practically a continuation of No. 1 Main slope. The other trends due north, or about parallel to the No. 4 slope. These three slopes open up a fairly large area.

*No. 2 Slope.* — The main slope was driven down a distance of 7,800 feet, at which point some poor coal was struck and development was stopped. Levels had been turned off to right and left as in No. 1 slope, the lowest being Nos. 20 East and West; but the only levels operating now are from Nos. 16 to 20 on the east side. A fault was encountered on the east side, and it cut off the coal. Prospecting along this fault was carried out in Nos. 15, 16, and 17 East levels, but the examination was evidently conducted in a different direction in No. 15 level from that followed in Nos. 16 and 17. In No. 15 the original lower seam was picked up, while in Nos. 16 and 17 another seam, referred to above as the Upper Bench and lying about 40 feet vertically above the lower seam, was found. This new seam was opened up and extensively worked in Nos. 16 and 17 levels. No. 15 East level, on the other hand, continued to operate in the original lower seam until a point was reached where it became necessary to leave a barrier pillar between it and No. 5 Mine lower seam workings. The levels below No. 17 were stopped when the fault was reached, and the pillars were withdrawn. No. 15 East level, which had been abandoned for some considerable time, was re-opened, and from it a slope trending due east was started inside the fault. On account of the thickness of the seam in this section, the pillar and stall system of mining was adopted. The slope is now down past the face of No. 17 level.

Another slope, also running due east, was started off No. 20 East level. A pillar 200 feet thick was left below the No. 20 East level, and then the longwall method was followed. The slope is now down 950 feet, and there are 24 places in the section. A new slope has also been started off No. 20 West level. It trends N. 45°E., or parallel to No. 2 slope, and it is now down 300 feet from the level. This slope will also be on the longwall system. Places are being turned off ready for operation.

These six slopes, three on No. 1 and three on No. 2, constitute the development scheme for the future, and they are giving access to territory hitherto untouched. At the same time, they have created serious problems in haulage and ventilation.

*Haulage.* — The haulage on Nos. 1 and 2 slopes is effected by a double drum hoist operated by a 500-h.p. motor. This is the same hoist which had been in use at the mine as a steam hoist. Both drums are loose and clutch-operated on a common shaft, so that hoisting may be done on both slopes at the same time or on either slope separately. There are 7,500 feet of rope on each drum. The main hoist on No. 1 slope operates to the No. 15 West level, a distance of 6,900 feet. From here a storage battery locomotive is used between No. 4 slope and the bottom of the main slope. Auxiliary hoists are used to raise cars from No. 20 level, and also from No. 4 slope, to No. 15 level, where they are picked up by the motor. Other auxiliary hoists are used to gather the coal from the new slopes below No. 20 level and place it on the landing at No. 20.

On No. 2 slope, the main hoist runs to No. 17 East level, a distance of 6,400 feet. All coal from below this point is raised by an auxiliary hoist on No. 17 level, whence it goes to the main surface landing.

Only two sizes of auxiliary hoist are used, 50 and 100 h.p. respectively. These hoists are all of the same make and are interchangeable.

Haulage at the face and on the levels is done by mules.

*Ventilation.* — Ventilation is probably the most serious problem confronting us in the operation of this mine. A Sullivan multi-blade fan, producing 165,000 cubic feet per minute, is used. This fan ventilates the entire mine. On account of the great distances the air has to be carried, and the number of stoppings required, the leakage naturally is considerable. To help reduce this leakage, stoppings are being rebuilt and "gunited" with the cement gun. In doing this, wire netting is first nailed to the face of the stopping, in order to give the required bond for the cement. It is possible to get a much better seal around the edge of the stopping by the "gunite" method than by any other process. Cracks and breaks in the strata are also readily sealed up in this way.

The return airways are being enlarged to permit the passage of larger quantities of air.

A Sirocco fan, which was used at No. 7 mine and which has a capacity of 200,000 cubic feet per minute, is being installed. When this installation is complete, it is proposed to make Nos. 1 and 2 slopes entirely separate so far as ventilation is concerned. The present fan will be used to ventilate No. 2 slope while the new one will ventilate No. 1. With the completion of this installation, it is expected that there will be more than sufficient air for all requirements.

*Pumping.* — There is not a great deal of water to handle in this mine. All pumps used are turbines, varying in size from single-stage to nine-stage. On account of the excessive length of slope and the amount of pipe required, two ten-inch bore-holes were put down and cased. The bore-hole on No. 1 slope is at No. 19 East level, that on No. 2 slope at No. 14 level. The water is pumped from the respective slopes up these bore-holes to the surface. The main pump on No. 1 slope is below No. 19 East level; it is a 7-stage pump with a capacity of 350 gallons per minute, and it is driven by a 150-h.p. motor, direct connected. The height of lift at this point is 780 feet vertical. In No. 2 slope the main pump is stationed at No. 20 West level and is a 9-stage pump with a capacity of 350 gallons, driven by a 200-h.p. motor. The vertical lift here is 1,180 feet. The auxiliary pumping is done by single and 3-stage units driven by 10 and 25 h.p. motors, respectively.

*No. 5 Mine.*

No. 5 mine is operating the upper seam. It is opened by a shaft. The shaft actually was sunk to the lower seam, but operations there were abandoned and the workings allowed to fill with water. The shaft intersects the upper seam at a depth of 279 feet. Here a main level has been driven to the east for a distance of 3,650 feet. Towards the west, however, a fault was encountered a few feet from the shaft and nothing further was done on this side of the shaft for many years.

The pitch of the seam is N.40°E. and the dip about 8 degrees. Two inclines, Nos. 1 and 2, were raised off the East level and worked by the pillar and stall method. These have now been worked out and the pillars extracted. No. 1 slope

was sunk at a point 1,050 feet from the shaft bottom. This slope was driven down 1,500 feet, at which point development work was stopped on account of the large amount of rock in the seam. Levels were turned off east and west. Those on the west side struck the fault running through the shaft bottom and those to the east were stopped on account of the excessive amount of rock in the seam. The average thickness of the seam is about 7 feet, but it includes three bands of rock. To the east these rock bands increase in thickness, and as the aggregate thickness of the seam remains about constant, the bands of coal become thinner and thinner until there is considerably more rock than coal in the seam.

No. 2 slope was driven down at a point 3,650 feet from the shaft bottom and was stopped 850 feet from the level. Here the thickness of the top bench of rock was nearly 4 feet so it was decided to open up the levels being driven east by longwall, making the thick band of rock the roof. The section now has 18 places working, and the longwall system is used throughout. Boreholes immediately ahead of the section show that there has been a displacement of the strata here. There are evidently two large faults just ahead of the present workings, and, within a short distance, they will entirely cut off this portion of this mine.

A tunnel was driven through the fault on the west side of the shaft and the seam located; its thickness, however, is much less than on the east side. Most of the rock bands are absent, and there is about 3½ feet of coal. Here a slope was driven, and it is still being developed, being now down a distance of 2,600 feet. Levels were turned off right and left. In the levels to the right the seam has about the normal thickness, and this section was worked by pillar and stall. To the left the coal is thinner and was opened by the longwall system. A regular network of faults was met with in the area, and they have considerably retarded the development of the levels. So numerous were they that it was impossible to open up any continuous section of face containing more than 3 or 4 places.

The faults have two general directions: N. 60°-70°E. and N. 10°W. They are sharp, well defined, and have displacements ranging from one foot to ten feet, and occasionally even more. The slope is at present being driven through a fault whose extent

has not yet been determined. The faulting has greatly retarded the development of the area, which would have been ideal for longwall operation had it been undisturbed. Although, as stated, the slope has been driven 2,600 feet, only a few working places are being operated on it.

*Mine Water.* — Next to the extensive faulting, the greatest problem in this mine is the handling of the water, which is present in large volume. Moreover, the water is comparatively acid, as the following analyses will show:

**Analyses of Water, No. 5 Mine.**

(Parts per million)

| | Shaft bottom, No. 5 mine | Top of No. 1 incline | Bottom of No. 1 dip | Bottom of No. 2 incline |
|---|---|---|---|---|
| Total solids | 5,205 | 3,713 | 2,381 | 4,862 |
| Organic and volatile matter | 117 | 227 | 128 | 274 |
| Sulphates, etc | 1,964 | 1,065 | 910 | 1,313 |
| Silica ($SiO_2$) | 85 | 63 | 33 | 180 |
| Ferric oxide ($Fe_2O_3$) plus small amounts of alumina ($Al_2O_3$) | 1,455 | 749 | 172 | 1,396 |
| Lime (CaO) | 372 | 376 | 356 | 446 |
| Magnesia (MgO) | 13 | 10 | 6 | 11 |
| Sulphur trioxide ($SO_3$) | 2,878 | 2,190 | 1,323 | 2,363 |
| Chlorides as Cl | 241 | 191 | 142 | 142 |
| Free acid as $H_2SO_4$ | 286 | 221 | 220 | 427 |

The acidity of the water is such that it rapidly attacks metals with which it comes in contact. Pumps of a special acid-resisting alloy were accordingly made in the company's own shops, but even with these the cost of maintenance was very high. Nos. 5 and 6 shafts being connected underground both in the upper and lower seams, it was decided to hoist the water by bucket from No. 6 shaft, which at this time was not producing coal. Self-filling and self-emptying cylindrical buckets, with a capacity of 600 gallons each, were used. Two hoists per minute can be made, giving a hoisting capacity of

1,200 gallons per minute. During the winter months, when the water is heaviest, this hoist is making two trips per minute and running, on an average, about 22 hours per day, which means a discharge of over one and a half million gallons of water per day. Three five-stage pumps are maintained at the bottom of No. 5 shaft for use in case of breakdown of the hoisting apparatus at No. 6 shaft.

Pumping to the shaft bottom from the various slopes is effected by 3- and 5-stage turbine pumps.

*Haulage.* — The main haulage on the East level is by electric trolley locomotive. A storage battery locomotive is used on the level from the bottom of No. 2 slope. Electric auxiliary hoists of the same size and type as those in No. 4 mine are used here. Haulage at the faces and short levels is done by mules.

## MINING METHODS.

In both mines the major portion of the mining is done by hand. Mining machines of the Sullivan type are in use in various sections of the mines where the conditions are at all suitable for machine mining. Both shortwall and longwall types of machine are employed. In No. 4 mine, the longwall sections in No. 20 East level and one of the sections of No. 4 slope, are being mined by machine. The adoption of machine mining throughout the mine is not possible, owing to the varying conditions and excessive faulting; the undisturbed area being in most cases too restricted to permit of the continuous operation of one machine.

## SCREENING ARRANGEMENTS.

Marcus horizontal screens are used at both mines. Coal is separated into two sizes, lump and slack. Slack is all coal which will pass through a plate with 2-in. perforations. The lump is hand-picked on the screen and when discharged from the latter is ready for the market. The slack is taken to a washery, and after washing is screened to three sizes, nut, pea, and slack. The finer grades are either sold separately, or they can be mixed with the lump to make a steam coal.

# ELECTRICALLY OPERATED COAL MINES.

## Equipment at the Mines of Canadian Collieries (Dunsmuir) Limited in the Comox District, Vancouver Island, B.C.

By Frank Sawford.

Annual General Meeting, British Columbian Division, Vancouver, February, 1922

The use of electricity in coal mining dates back to the earliest development of electrical equipment. Just as soon as it had been demonstrated that electricity could be utilized commercially for signalling, lighting, communication, transportation, motive power, etc., its adoption for such purposes in coal mining immediately followed. The art of coal mining is at best a difficult one, and those responsible for the operation of the mines are ever on the lookout for new devices or methods making for increased economy and efficiency, and which at the same time lessen hazard to life and property. In the furtherance of these endeavours the introduction of electrical equipment has played an important part. Conversely, the experience gained through the use of such equipment in coal mines has contributed very materially towards bringing electrical apparatus and machinery to its present high state of development.

It cannot be said, however, that all coal mines are equipped with the most modern appliances — and this is true not only in British Columbia but of coal mines in almost any part of the world. Even in the same district the greatest extremes are to be met with. The main reason for this is that coal mining is, in many places, a most erratic and uncertain business, so that it is frequently impossible to plan methods of operation sufficiently in advance of development to permit of the most efficient equipment being installed at the outset. This is particularly true in British Columbia, where the coal measures are generally much disturbed, and where even a liberal use of the diamond drill does not always afford the mine operator a sure guide to the probable future development of the mine.

As a consequence, the decision as to what may constitute the most efficient type of equipment, electrical or otherwise, in any particular case, calls for very careful consideration.

Generally speaking, operators in the United States, and in Great Britain and continental Europe, are not faced with these problems, or at least not to the same extent. They work in coal measures which, as a rule, are comparatively uniform, and the mines can be laid out, and plans made in advance, to utilize the most modern and economical equipment available. In view of these circumstances, charges of laxity and inefficiency such as are sometimes laid against British Columbian operators are hardly justified. Notwithstanding all the difficulties that have to be contended with, there are many coal mines in British Columbia which compare favourably with similar mines in any part of the world in the matter of modern electrical equipment. In the present paper it is the intention to describe this equipment in the group of mines operated by the Canadian Collieries (Dunsmuir) Limited, in the Comox district, Vancouver Island. Since 1913, these mines have been equipped throughout for electrical operation.

When, some years ago, the Canadian Collieries took over the mines formerly operated by the Dunsmuir interests, they became possessed of an assortment of equipment of the most antiquated, obsolete, and inefficient type. This is no reflection on the former operators. These mines, like others which are operating in the disturbed and faulted coal fields of Vancouver Island, had perforce been opened up with more or less temporary equipment, and, as the mines were developed, this was re-arranged and added to until it had grown to relatively large proportions. So long as plant is being built up in this way, operators are naturally reluctant to discard any part of it, provided it will work. On the other hand, new operators taking over such a property, already proved and developed, have every inducement to scrap old and out-of-date equipment and replace it with a modern and efficient plant in order that costs may be reduced to a minimum and the output maintained or increased. This was the course decided upon by Canadian Collieries.

There were at the time four mine openings in the Comox field, all producing coal, and it was desired to open a fifth

in order to increase the output. The four operating mines were known as Nos. 4, 5, 6, and 7. All were equipped with plant of a nondescript nature. Power was generated by steam in hand-fired boilers of various sizes and types—boilers thirty years old were found coupled with boilers only a few years old, resulting in compulsory operation at low pressures. More than ten per cent of the output of the mines was used for raising steam for the operation of the plants, and even at that the output was curtailed for lack of power.

Long steam lines were in use, both overhead and buried in the ground, and all without any attempt at insulation. As a consequence, the drop in steam pressure, due to condensation, was everywhere serious and in some lines very considerable. In one place, for example, a fan engine was found operating at 20 pounds at the throttle while the boiler pressure was 80 pounds. It was only possible to operate the fan by running the engine with the cylinder cocks and steam line drains wide open all the time, to permit water of condensation to escape.

The former operators of the mines were evidently not satisfied with the low efficiency of the steam generated power. At No. 4 mine an attempt had been made to introduce electrical transmission for the operation of the pumps underground. Unfortunately, the experiment was somewhat crude. The plant consisted of three horizontal piston valve engines, driving direct current 250-volt generators by belts. The generators were of antiquated types, one being an Edison No. 48, built by the old Edison Machine Works before the formation of the General Electric Company. There was a total of about 180 k.w. in generating equipment, handling pumps aggregating approximately 100 h.p., operating at about 50% load factor. Small wonder that the performance of this installation was such that electrical transmission was not considered to have the advantages claimed for it over transmission by steam or compressed air.

After a careful survey of the situation, it was decided by Canadian Collieries to scrap practically the whole of the old plant, retaining only such portions of the equipment as were in fairly good shape or which could be adapted to efficient working.

A scheme for the installation of new steam plants at each mine was considered but was rejected in favour of the adoption of electrical transmission throughout, both for surface work and underground. One factor influencing this decision was the excessive labour costs incidental to steam plants, which would have had to be practically duplicated, with complete crews of engineers and firemen, at each of the several mines.

### THE CENTRAL HYDRO-ELECTRIC POWER PLANT.

A central power plant having been decided upon, the next consideration was as to whether this should be hydro-electric or steam. Water power of ample capacity, and suitable for good hydro-electric development, was available. Careful investigation showed that while the estimated initial cost was slightly higher for hydro-electric development than for a steam plant, operating and maintenance costs, including interest and depreciation, were very much lower. Accordingly, the choice fell on the hydro-electric plant, and, after more than eight years' operation, it is gratifying to note that the low estimated operating and maintenance costs have not been exceeded, in spite of the low plant load factor.

The source of the water is Comox lake. Here, at the outlet from the lake into Puntledge river, a reinforced concrete impounding dam, 300 feet long and 20 feet high, was built, together with the usual controlling gates and spillways. The outlet channel was lowered about six feet to allow a larger draw-off from the lake. On Puntledge river, about a mile below the impounding dam, a diversion dam of the weir type was built, also of reinforced concrete. This dam allowed the maximum flow of the river and maintained the necessary level for diverting the required water from the stream by means of vertical sluice gates.

From the diversion dam the water flows through a canal, 330 feet long, and thence along a semi-circular flume, 12 feet wide and 2,200 feet long, built of wood staves and suspended from a trestle. This empties into a concrete lined ditch, 900 feet long, which conveys the water to the forebay. The latter is about three and a half miles distant from, and at an elevation of 350 feet above, the power house, which is situated on the banks of Puntledge river.

Throughout almost their entire course from the forebay to the power house, the pipe-lines and penstocks are constructed of wood stave pipe. The pipe has a diameter of 8 feet for the first 5,400 feet. At this point a reinforced concrete Y-branch is introduced, providing for connection, on the down-stream side, with two 6-foot pipes. Up to the present, only one of the latter has been installed. Six-foot gate valves are fitted to the openings of the Y-branch.

The 6-foot pipe extends for a distance of 4,500 feet and ends in a second Y-branch, which has two 54-inch gate valves on the down-stream side. Provision has been made here for the installation of another similar Y-branch to receive the second 6-foot pipe line which may, at some future time, be laid alongside the present one when it becomes necessary to increase the generating capacity at the power house. From this Y-branch, two 54-inch pipes, still wood stave, extend for a distance of 3,100 feet, but from this point to the power house, some 660 feet, lock bar steel pipe is installed, as in this section the head is too high for the economical use of wood stave pipe.

The pipe line is fitted where required with air valves, in clusters of four. The installation was designed for an ultimate flow of about 800 cubic feet per second, each 54-inch pipe having a capacity of 190 cubic feet.

At the entrance to the power house the two 54-inch penstokes are fitted with Pitot tubes (General Electric type) for measuring the flow of water. A portable mercury manometer is used as a flow meter.

The power house proper measures about 120 by 30 feet, and adjoining it is a three story wing, 50 by 20 feet. In the main portion are the turbines and generators, while the wing houses the switching and electrical equipment. The building is very substantially constructed of reinforced concrete. A 40-ton hand power travelling crane spans the main bay.

The generating equipment consists of two Escher-Wyss Francis turbines of 6,000 h.p. each, direct connected to two Canadian General Electric 4,400 k.w. generators, which include 50 k.w. 125 volt exciters. The combined sets operate at a speed of 500 r.p.m. The turbines are designed to develop their full rated power at a working head of 270 feet, and are

controlled by oil pressure governors, operating on the wicket gate. Power for operating the oil pressure pumps on the governors is supplied by small direct connected impulse wheels. Each turbine is also fitted with a special relief or by-pass valve, connected to the governor and operating in synchronism with the wicket gates. In order to maintain a steady flow of water, and to prevent shocks or surges in the pipe lines, these relief valves are fitted with dashpots for slow closing and also to provide a means of adjustment to vary the quantity of water flow as required, depending on the maximum peak loads in the turbines.

Sufficient centrifugal energy to control the turbine speed on a varying load in accordance with the pipe line characteristics is supplied by means of a flywheel mounted between the turbine and generator.

The generators are of the revolving field type, and very solidly constructed. They successfully passed a runaway overspeed test, proving that they are perfectly safe in case of a sudden removal of the load, and governor failure. They operate at 13,200 volts, 3 phase, 25 cycles, and have a liberal overload rating. The end windings are closed in by means of sheet steel shields, and fan blades are fitted to the solid rotor centres. The latter are pierced with ventilating ducts, which, with the end shields and fan blades, provide ample ventilation. The connecting cables from the generators to the switchgear are fabric insulated, lead covered, and drawn in fibre ducts buried in the floor and walls as required.

The remote control switchboard is mounted on a gallery overlooking the main bay of the power house, the back of the control board facing into the switchgear annex. The control gear is of the C.G.E. type, and contains two exciter panels, two generator panels, two spare blank generator panels, and five feeder panels. Four of the latter supply the outside transmission lines while the fifth supplies the station. A Tyrrell regulator is also provided for control of the voltage, and is arranged for parallel operation of the exciters.

In explanation of the two spare generator panels referred to above, it should be stated that the station is designed for two additional generating sets, thus doubling the capacity of the plant when required. This expansion would be effected

by the removal of the end wall of the main bay, and, when this addition is built, the switching annex will occupy the central portion of the machine bay.

The switching annex is arranged in three floors. The bottom floor houses one 100 k.w. station transformer, with the ratio of 13,200 volts to 110 volts; the instrument transformers are arranged on the walls and ceiling, while the rest of the floor is used as a repair shop and store room. On the second floor are the main oil switches and bus bars, all mounted in reinforced concrete cells. The main oil switches are C.G.E. motor operated; each pole is enclosed in a separate cell. The third floor contains four sets of C.G.E. electrolytic lightning arresters, connected to the four outgoing transmission lines, the outlets for which are arranged through the walls of this room.

In designing this power house, the underlying idea has been to make it as simple as possible. The absence of step-up transformers, with their necessary switchgear, has contributed greatly to the attainment of this object. Also, no apparatus is used which is not strictly necessary for efficient operation. As an illustration of this, it may be stated that there are only two recording instruments in the station, and these are pressure pumps, for recording the pipe line pressures. The low operating and maintenance cost of the station are striking testimony to the advantages of this simplicity in design.

Although the entire station is fireproof, it is plentifully supplied with hydrants. Heating throughout is electrical.

This power house may well be cited as a good example of modern equipment and excellent engineering applied to coal mining. During its eight years of operation, practically no interruptions in supply have occurred.

## THE TRANSMISSION SYSTEM AND SUB-STATIONS.

The electrical transmission system is deserving of notice. A double 3-phase 3-wire circuit leaves the power house as a separate power line and ends at a sub-station at No. 7 mine. The sub-station equipment and method of switching the lines will be referred to later. From No. 7 mine a single 3-phase

3-wire circuit is carried to a sub-station at No. 4 mine, and thence a similar circuit leads to a sub-station at No. 5 mine. From the latter a double 3-phase circuit is carried to a sub-station at No. 8 mine, from which one double circuit pole line leads back to the power house, and one single circuit pole line to a sub-station at Union Bay. It will thus be seen that the transmission line forms a complete ring of the mines, additional to which is the line from No. 8 mine to Union Bay. Altogether, there are about 35 miles of 3-wire 3-phase transmission line.

The construction of the pole lines is very substantial. The poles are of cedar, with a minimum length of 40 feet and at least seven inches across at the top. Where necessary they are braced — this being considered preferable to the use of guys. The cross-arms are 1¾ in. locust, and pole-top pins are oak with a 2-in. square shank which is bolted through the pole top. Insulators are brown glazed porcelain, Locke No. 101. Throughout the line, the wire used is No.O. B.&S. stranded aluminium cable, and all splices and tap-offs are made with aluminium sleeves.

No ground line wire is used, owing to the fact that, the district being heavily timbered, atmospheric disturbances are not very severe. Also, ample lightning protection is provided at each section of the system.

A metallic circuit telephone line is carried along the pole line, being attached to the poles, five feet below the lower cross-arm, by means of ordinary side blocks, on which are mounted 11,000-volt porcelain insulators. Very little inductive interference is experienced. The system connects all the sub-stations and the power house, the telephones being bridged across the lines and fitted with protective transformers, arresters, and fuses specially designed for such installations. The telephone line wires are transposed at every third pole, and the power lines are also transposed, each section having a complete turn throughout its length, the transpositions being equally divided in the length of each section.

Where necessary, the telephones in the sub-stations are fitted with extension bells, and are placed in sound-proof booths. There is an entirely separate telephone system connecting the mines.

Except as regards capacity, the sub-stations at the several mines and at Union Bay are duplicates of one another in arrangement and design. Each building houses, not only the electrical equipment, but also the mine hoisting and other machinery. This eliminates the necessity of having a special sub-station attendant, the hoisting engineer doing whatever switching is required.

For the high tension wiring inside the building, ¾-in. light gauge copper tubing is used, supported on porcelain line insulators. Special fittings have been designed to furnish supports for the insulator and clamps for the copper tubing.

The high tension leads pass through disconnecting switches set in concrete barriers, to non-automatic Westinghouse oil switches, and then up the other side of the cell structure through a second set of disconnecting switches to a series of bus bars run horizontally along the top of the cell structure. High tension leads to the primary of the 13,200-volt transformers are also tapped from one end of the bus bars, passing through sets of disconnecting switches, Westinghouse oil circuit breakers, and current transformers, overhead to the transformer primaries.

Choke coils are inserted in the bus bars between the line switches and the transformer circuit breaker, while, at the end of the bus bars nearest the line switches, a set of electrolytic lightning arresters is connected. By connecting the arresters to the bus bars in this manner inside the building, one set of arresters takes care of the whole station, obviating the necessity of duplicating these sets for each incoming and outgoing line.

The step-down transformers are all of the Westinghouse oil insulated, water cooled, 3-phase type, with 13,200-volt primaries and 2,200-volt secondaries. There is also provided at each station a similar transformer with 2,200-volt primary and 440-volt secondary. All constant speed motors of 50 h.p. and over are operated on the 2,200-volt service; variable speed motors, such as are used for hoisting and similar work underground, as well as all constant speed motors below 50 h.p., are taken care of by the 440-volt service.

Sub-stations Nos. 4, 5 and 7 each contain one 1,500 k.w. step-down transformer, and one 500 k.w. transformer for

440-volt service; No. 8 is equipped with two 1,000 k.w., and one 500 k.w., machines; and at the Union Bay station there is one 500 k.w. step-down transformer, and one 250 k.w. machine for 440-volt service.

All transformers are mounted on trucks with flanged wheels and these stand on short tracks which lead through large double doors placed in the wall immediately opposite each transformer. By this means a defective machine, which might cause a fire, can be quickly run out of the building. Large wire cutters, with insulated handles, are kept in each station for the purpose of cutting away the transformer connections in case of emergency.

In all the stations, the low tension switchboard for 2,200-volt service is placed at the opposite side of the building from the step-down transformers. This arrangement is very effective in simplifying the wiring, and also has the advantage of keeping the high tension wiring entirely apart from the low tension wiring or switchgear. The high tension switches are operated mechanically by remote control panels, arranged so as to form an alleyway, about six feet wide, with the high tension control panels on one side and the low tension 2,200-volt service switchboard on the other side. This arrangement of the switchgear is modified in some of the sub-stations to suit the available layout. Low tension wiring in the stations is laid in steel conduits wherever possible.

On the 2,200-volt service switchboard, provision is made for separate switching and metering for surface operations and for underground pumping, hoisting, ventilating, and general service. Integrating poly-phase meters, which are installed in addition to the usual indicating meters, make it possible to ascertain the amount of power consumed in each service. Also, a swinging bracket at the end of the switchboard carries a power factor meter and voltmeter.

A daily log is kept at each sub-station, and also at the power house; and monthly summaries are made, from which can be computed the consumption of power, per ton of coal mined, for each of the various services, such as hoisting, pumping, ventilating, etc.

It may be thought that the system is somewhat elaborate for coal mining operations. In reality it is very simple, and

yet every provision has been made for uninterrupted service, which is essential in a coal mine. That the general arrangement of the system has justified itself in every way is shown by the fact that, in eight years of operation, very few interruptions have occurred.

Each mine is charged with the power it consumes as shown by the integrating meters in the respective sub-stations. The difference between the total of the sub-station readings and the power house readings is charged against transmission losses. In this way the power house is credited only with the power actually delivered to the sub-stations. The power is charged to the mines at a flat rate of one cent per kilowatt hour. Accounts for the power plant and the mines are kept entirely separate.

Besides taking care of its own mines and plant, the company also supplies power in bulk to the cities of Cumberland and Courtenay, and to the towns of Royston and Union Bay.

The foregoing description covers the main power house and the sub-stations, and may therefore be classed as the power production and distribution end of the scheme. From the point of view of coal mine managers and operators, perhaps the most interesting part of the system is the actual application of the power at the mines, or the power using end of the scheme.

## No. 8 Mine.

When the scheme for the electrical operation of the mines was first launched, the intention was to put in the most complete installation at No. 8 mine. Shortly afterwards, however, and before the mine was developed to any great extent, it was completely closed down. But equipment including two electric hoists had already been installed, and as these hoists were among the largest of their type in Canada, a brief description of them may be of interest.

The main shaft hoist is of the Ward-Leonard type. It was designed for working at two levels, the 700-ft. and the 1,000-ft., with a capacity of 2,070 tons in eight hours when hoisting from the lower level, and with two cars per cage, balanced working.

The drums are of the cylindro-conical type, grooved for 1¼-in. rope, and with diameter tapering from 12 ft. to 7 ft.

Rope speed averaged 2,500 feet per minute, with a maximum of 3,150 feet per minute. The total winding time was 33½ seconds, or, moving from deck to deck, 15 seconds.

The winding motor, which, of course, was direct coupled to the hoisting drum shaft, has a maximum rating of 1,400 h.p., and the average load was 750 h.p., It was fitted with commutating poles, and Siemens compensating windings. The normal speed was 83.8 r.p.m. at 560 volts direct current. The shunt field was excited at 220 volts direct current.

This motor has an efficiency of 91%. It, and the other electrical equipment, was furnished by Siemens & Co., of England. The mechanical equipment, now to be described, came from Robey & Co., also of England.

The two conical drums are of cast steel in the conical portion, with cast iron sides in halves. Each drum is loose on the shaft, and fitted with bronze bushings. The shaft is 14 inches in diameter. Each drum is fitted with a post brake operated by compressed air, the brake engines being of the floating type and controlled from the driver's platform. The brakes are equipped to hold the drums against 150% of the motor torque. The drums have Hall friction clutches, also operated by compressed air engines of the floating type, controlled from the driver's platform.

The clutch and brake levers are interlocked, so that neither clutch can be released unless its corresponding brake is hard on. Clutches are of sufficient capacity to drive, without slipping, a torque equivalent to 150% of the maximum rating of the motor.

The hoist is also fitted with a pair of car haul drums, gear driven from the main shaft. These drums have change gears to enable them to haul 50 feet of rope in the same time that the main hoist winds either 600 or 1,000 feet. As their name implies, they were for hauling empty cars on the tipple back of the caging level.

The equipment also includes a vertical depth indicator, showing the depth independently on both drums; safety

cams, to prevent over-winding (and, as an additional safeguard, limit switches above the shaft cages); and a Karlik tachograph, for recording the complete cycle of each wind.

On the pit head, within convenient reach of the cager, is a switch for limiting the speed of the hoist. This, being under the control of the cager, prevents the hoist operator from hoisting at full speed when men are being brought to the surface.

There is a separate motor-driven air compressor, with receiver and automatic switch, to furnish air at 110 pounds pressure for control of the brake and clutch engines. The control levers are equipped with a pressure device which prevents the operation of the hoist unless sufficient air pressure is available for the proper working of the clutch and brake engines. The levers are also locked against operation unless electrical power is available for full excitation of the motor field.

There are several other safety devices, which with those described, provide against every probable contingency, render the hoist practically fool-proof, and make it possible to operate at high speed with perfect safety.

The motor generator set by which the hoist is controlled consists of three machines — motor, control dynamo, and exciter — which are mounted on a common bed-plate. They are direct coupled, and operate at 740 r.p.m. The driving motor is of the slip-ring type, 750-h.p., 3-phase, 25-cycle, 2,200 volts, with an efficiency of 92%; the direct current control dynamo is 600 k.w., 500 volts, and is fitted with commutating poles and Siemens compensating windings; and the exciter is 24 k.w., 220 volts. The motor is started by means of a liquid rheostat.

The complete electrical equipment for the Ward-Leonard control system consists of three principal parts; (1) winding motor, (2) motor generator set, and (3) control gear. A brief description of the operation of the system may be of interest.

The direct current winding motor is shunt wound and constantly excited from a separate source. As a result, the speed is exactly proportional to the armature voltage and quite independent of the load on the motor. Thus the speed at which

the motor runs, and therefore also the winding speed, is independent of the load being hoisted.

The motor receives current from a direct current dynamo, known as the "control dynamo". The shunt field of this latter is also excited from an independent source, and, by a suitable arrangement, the field can be regulated from zero to full excitation in either direction. Since the speed of this dynamo is constant, the pressure at its terminals, and thus the voltage at the armature of the winding motor, is exactly proportional to the field current; and, as the speed of the winding motor exactly corresponds to its armature voltage, the winding speed must exactly correspond to the excitation of the control dynamo.

If the direction of the exciting current is reversed, the armature current of the winding motor, and thus, also, the direction of rotation, is reversed — this because the winding motor is shunt wound but separately excited at a constant value.

The regulator for the shunt field is directly connected with the driver's control handle, so that the speed of the winding motor corresponds exactly to the position of this handle.

It will be clear that, if the control dynamo is fully excited, and thus the winding motor running at full speed, on reducing the field strength the pressure at the terminals of the control dynamo will fall. Immediately this occurs, the momentum of the load in the shaft, and of the revolving parts, tends to keep the winding motor running at full speed. The voltage of the control dynamo having dropped, however, that of the winding motor will be the greater; the latter, therefore, will act as a generator and the control dynamo as a motor, giving a powerful electric breaking action until the system is once more balanced and the winding speed has adjusted itself to the new conditions — which, of course, is practically instantaneous.

From the foregoing description, it will be seen that this system of control is ideal for high-speed shaft hoisting. It is used extensively in Great Britain and continental Europe, where deep shafts are common. In many cases a modification is introduced by the addition of a heavy flywheel to the motor

generator set — this is the Ilgner system. The object of the flywheel is to smooth out the power peaks on the generating plant when the hoist is being accelerated or retarded. In the installation at No. 8 mine of Canadian Collieries, however, the Ilgner system was regarded as unnecessary, on account of the relatively large size of the generating units.

The second hoist at this mine was installed at the air shaft. It is of smaller capacity than the main shaft hoist, and was designed principally to supplement the latter in hoisting and lowering men, and also to relieve it of handling timber and supplies.

This hoist is equipped with a plain cylindrical drum, 6½ ft. in diameter by 7½ ft. wide. It was designed for working at the 700-ft. and 1,000-ft. levels, with a capacity of 429 long tons in 8 hours from the lower level, single cage and single car working, and winding unbalanced. The maximum rope speed is 1,500 feet per minute. The winding time was 43.5 seconds, and banking 10 seconds.

The winding motor has a maximum power of 500 h.p., and an average of 375 h.p. at 74 r.p.m. working at 550 volts with an efficiency of 90.5%.

Except in size, the electrical and mechanical equipment of this hoist is similar to that of the main hoist.

Other equipment at No. 8 mine included the all-steel tipple and head-frame. In the tipple, two pairs of Marcus screens and picking tables, operated by 440-volt induction motors, were installed; also a 3-car gravity dump and other necessary machinery, such as quick acting scales, automatic rock car dump, etc. The arrangement of the screens and picking tables permitted of the coal being graded into several sizes for more convenient picking, and the several sizes could be re-combined or left separate as desired.

## No. 7 Mine.

The equipment at the No. 7 mine differs from that at No. 8. The mine being worked by a slope, the Ward-Leonard system of hoisting was not necessary. The hoist, or rather

the haulage machine, installed was designed for hauling 40 tons per trip up a slope with an average grade of 10% and a rope speed of 1,320 feet per minute. The main drum is 8 ft. in diameter, with a capacity of 9,000 ft. of 1¼ in. rope. It is loose on the shaft, and is driven by a heavy band clutch of the "Lame" type. A very heavy band brake is provided. Both clutch and brake are operated by compressed air engines with cataract cylinders, the operating valves being arranged with a relay mechanism for floating control.

The hoist is driven by a 750 h.p. induction motor, operating at 250 r.p.m. The motor is connected by means of a Bruce-Macbeth flexible coupling to a jackshaft on the hoist, and this in turn is connected to the drum by a single reduction of cast steel herring-bone gears, enclosed in a gear case. The hoist is equipped with a depth indicator and a recorder for recording the hoisting cycles.

The motor is operated by means of a liquid type controller — this is believed to have been one of the first applications in Canada of this type of controller for operating mine hoists.

The stator of the motor is operated at 2,200 volts, and reversing switches of the air brake type are used for reversing the direction of rotation. The liquid controller is connected in the rotor circuit. Stator, reversing switches, and controller are all operated by a single lever mounted on the driver's platform alongside the brake and clutch levers.

The liquid controller consists of two tanks. The lower and larger one contains water coils for cooling the electrolyte. Mounted above this is a smaller tank, the rheostat or controller, which contains a large number of steel plates suspended vertically. The electrolyte is pumped from the lower to the upper tank by a motor-driven centrifugal pump, which runs continuously while the hoist is in operation, circulating the electrolyte through the upper tank, whence it returns to the lower one through a short length of large diameter pipe.

This pipe, which is in reality a moveable weir, is suspended from a system of levers mounted on a horizontal shaft. The latter is driven through a worm reduction gear by a small reversible motor. When the weir is at its lowest, only the

tips of the plates are immersed in the circulating electrolyte; but as the weir is raised by the control motor, more and more of the plate surface becomes submerged, and there is a corresponding reduction in the resistance of the rotor circuit of the hoisting motor and an increase in its speed and torque. The small control motor is operated from the driver's platform by the control lever already referred to, through an electrical relay mechanism which determines the movement of the control motor in such a manner that a definite position of the control lever gives a corresponding position of the weir.

While this method of control is not as sensitive or accurate as the Ward-Leonard system, it is extremely satisfactory for the class of service required at No. 7 and similar mines.

The hoist at No. 7 mine is installed in the same building as the power sub-station. The equipment here includes an Ingersoll-Rand compressor (a piece of the old equipment) belted to a 500-h.p. synchronous motor. A synchronous motor was selected for this work because it offered a continuous load, and as some means of power factor correction was desired at this sub-station, this compressor drive was deemed to be the most satisfactory.

The motor operates at 500 r.p.m., the belt being 42 in. wide and running about 4,500 feet per minute. It was started up as an induction motor on reduced voltage, by auto transformers, the running voltage being 2,200. The exciter is directly connected, the whole machine being mounted on slide rails for tightening the belt. It was found in practice that the power factor could be corrected to unity by this machine.

The ventilating fan at this mine is a reversible Sirocco of 270,000 cubic feet capacity at 5-in. water gauge, running at 245 r.p.m. It is driven by a direct connected slip-ring motor of 350 h.p., connected by means of a rubber cushion flexible coupling. This motor is started by an automatic liquid rheostat, specially designed for the purpose, and as this application is believed to be the first of its kind, a brief description may prove of interest.

The rheostat, or starter, is entirely automatic; that is, as soon as power is applied to the starter terminals at the cor-

rect voltage the motor is immediately started and brought up to full speed, and continues in operation. Should the power supply have failed for any reason, the motor, on the resumption of the supply, at once goes into operation without the need of an attendant. As the fan is situated at some distance from the sub-station, much time is saved in the event of power interruptions.

It was originally planned to use synchronous motors for driving the fans at all the mines, in order to obtain power factor correction; but at the time no manufacturers of electrical equipment would undertake to furnish automatic starters for synchronous motors in this service. Rather than eliminate the automatic feature, which was considered most important, it was decided to use induction motors instead, and to obtain power factor correction by other drives. Hence the synchronous motor on the compressor at No. 7 mine.

The liquid rheostat is connected across the slip ring of the motor, the plates being mounted on the shaft, which is driven through a worm gear by a small motor. Power at 2,200 volts is supplied to the stator of the motor by oil immersed contactors, the operating magnets of which are controlled by a small pilot switch. Relays are connected through auxiliary contacts on the main contactors, which operate the pilot motor on the rheostat. When power is applied to the system with the rheostat open, the main contactors close, applying power to the stator of the main motor. At the same time, the circuit is closed to the pilot motor, which starts up and winds the rheostat plates into full immersion in the electrolyte, bringing the main motor up to full speed in about 30 seconds. As soon as the plates are fully immersed, they are short circuited by knife-blade contacts and the pilot motor stops. If power is interrupted, the main motor of course stops, and the main contacts drop out, setting the relay contacts in a reversing position for the pilot motor. When power is resumed, the main contactors cannot close until the pilot motor has wound the liquid rheostat to the open position, at which point a contact is made, which closes the main contactors, and this in turn reverses the pilot motor, returning the liquid rheostat to the full-on position and thus starting up the main motor.

The equipment in this sub-station includes a recording gauge, fitted with high and low alarm contacts, for recording the water gauge developed by the fan.

Marcus screens, driven by small induction motors, were installed in the tipple.

Underground, the electrical equipment of this mine was not very extensive, on account of compressed air being used — this is the only mine in the group in which compressed air was retained. Several motor driven turbine pumps and hoists were in use underground, however, at points where compressed air was not available or not satisfactory.

## No. 4 MINE.

The main hoisting equipment at No. 4 mine differs from that at No. 7 or No. 8 mines. The old steam driven hoist, designed and built by the Danville Machine Works, consisted of a twin cylinder, balance slide, valve engine, direct connected to a 4-drum hoist. It was designed for slope working in two separate slopes, and also for tail rope working. All four drums were mounted loose on their shafts and were clutch driven.

The main hoisting drums were mounted side by side on the first motion shaft, on which were two crank discs, set at 90 degrees, to which were coupled the engine connecting rods. The tail rope drums were mounted in front on a similar shaft, the two crank discs of which were connected to the main drum-cranks by side connecting rods.

This hoisting engine was retained, and converted to electric driving by the simple expedient of removing the engine connecting rods, mounting a split steel cut herring-bone gear on the main hoisting shaft, meshing into a cut steel herring-bone pinion mounted on a jackshaft, and connecting the latter by means of a flexible coupling to a 500-h.p. slip-ring induction motor.

The motor was controlled by a liquid type controller similar to that at No. 7 mine, excepting that the weir control

in the rheostat was manually operated from the operator's lever instead of by a pilot motor.

This installation affords an excellent example of the conversion of an existing steam hoist to electric driving without interruption to the operation of the mine, and at a low cost, since standard equipment was used for the electrical features.

Excepting that the fan is of a different make, the ventilating equipment at this mine is practically identical with that at No. 7. The electrical equipment, including the automatic starter, is exactly similar at the two mines.

Small motors, of various capacities, which were installed at this mine for operating the tipple, for pumping, etc., call for no special comment. The underground equipment will be referred to later.

## No. 5 Mine.

The hoist at No. 5 mine has a single drum, 7 ft. in diameter by 5½ ft. long, grooved for 1¼-in. rope. The shaft is 300 feet deep, and the capacity of the hoist is 7,200 lb. at 1,100 ft. per minute. The hoisting cycle is: acceleration, 12 sec.; full speed hoisting, 7¾ sec.; retardation, 5 sec.; and brakeing, 8 sec.; total time, 32¾ seconds.

The drum is keyed to the shaft and fitted with post brake, and is driven through single reduction herring-bone gearing by a 300-h.p. slip-ring motor. The brake is air operated with floating control. The installation includes also a large depth indicator, a safety controller for the prevention of overwinding, and a tachograph for recording the winds made.

The motor is rated at 300 h.p., 2,200 volts, 3 phase, 25 cycles at 245 r.p.m.; it is connected to the hoist by means of a Bruce-Macbeth flexible coupling. The controller is of the liquid type — a duplicate of that used on the hoist at No. 4 mine.

It will be noted that although this is a shaft hoist, a slip-ring induction motor was used rather than the Ward-Leonard

system of control as employed at No. 8 mine. It was considered that the additional expense of the latter system was not justified here as the shaft is only 300 feet deep, as against 1,000 feet at No. 8 mine.

No. 5 mine is ventilated by a Sirocco fan similar to that at No. 7. The motor and control are also an exact duplicate of the equipment at No. 7 mine. The tipple is operated by small motors, and calls for no special comment.

## No. 6 Mine.

No coal is now produced from No. 6 mine, but the shaft is used for hoisting water. The mine has no sub-station. It is supplied with power at 2,200 volts from No. 5 sub-station, from which it is distant only about one mile. The hoist is operated by a 250 h.p. induction motor connected with the old steam hoist — the equipment being similar to that at No. 4 mine, except that the hoist was a first motion double drum type without clutches. A herring-bone gear was bolted to one of the crank discs, and the motor was mounted on a concrete base, with a suitable herring-bone pinion, giving the correct hoisting speed ratio. As at No. 5 mine, the shaft is 300 feet deep. The motor is equipped with a liquid controller like that at No. 5 mine. This hoist un-waters both No. 5 and No. 6 mines, which are connected underground, and it has been very successful.

Underground pumps are maintained at No. 5 mine, but the hoist is used in preference on account of the mine water being acid, which causes rather heavy maintenance on the pumps. The hoist also has a slightly higher efficiency, and a very low maintenance cost. The pumps are therefore held in reserve, in case of emergency.

## Union Bay.

Union Bay, the shipping point of the company, is fed by a single transmission line about 13 miles long. Here are located the machine shops, carpenter, boiler, and blacksmith

shops, and foundry, as well as a coal washer, laboratory, and other incidental buildings.

Everything at Union Bay is electrically operated by individual motors — the loading conveyors, coal washer, shops, etc. In addition, several motor-driven pumping sets are in use, furnishing fresh water, salt water for coal washing, and water for fire protection.

## UNDERGROUND EQUIPMENT.

At No. 4 mine the power is transmitted to the underground workings by three-core rubber insulated armoured cable, which drops vertically through a cased borehole to a depth of 800 feet. The entire weight is supported on a concrete pier by means of a special clamp which grips the outer armouring, the conductors being brought through the centre of the clamp and sealed in a cast-iron box. Connection is made from this box to the transmission pole line. The cable and suspension system were specially designed for this class of work. The lower end of the cable is connected through an oil circuit breaker, and from this to the distribution system underground and the necessary stepdown transformers.

For the large pump motors the power is used direct at 2,200 volts; for the smaller pumps, hoists, coal cutters, etc., the voltage is reduced to 440.

The lighting generally is by means of incandescent lamps, installed in gas proof globes and fittings, the wiring being enclosed in steel conduit or lead covered cable.

Perhaps the least satisfactory feature of the underground equipment is the 2,200-volt distribution. On account of the long distances concerned, the cost of the armoured cable for this work is practically prohibitive, and in addition to this a large number of junction boxes have to be provided and the work is of a very special nature. Moreover, frequent changes are required in the route of the cable system. It was therefore decided to use standard 3,000-volt rubber-covered cable mounted on split porcelain insulators, these being fastened to

the mine timbers as much out of the way as possible. While not ideal, this method at least has the merit of low first cost and low maintenance cost, and, so far as is known, it has not given any serious trouble.

The borehole cable at No. 4 mine was the first to be installed at any of the company's mines. Later, when estimates were called for cables of similar construction for other of the mines, it was found that the price had risen to a prohibitive figure, while delivery could not be guaranteed within a reasonable time. It was accordingly decided to try the experiment of building up a cable from such materials as were readily available.

An ordinary ½-in. 6 - 19 strand galvanized wire rope was used as a core. This was served with a layer of tarred marlin, wrapped close, and painted with bitumen compound. Over this were laid the three conductors, spaced at 120 degrees around the central ore. The conductors were ordinary rubber covered 3,000-volt cable. The whole was then served with another layer of tarred marlin, wound close, and painted with bitumen compound. The whole process was carried out on the spot, and the cable lowered into the borehole as completed, the wire rope and the conductors being passed over pulleys mounted on a temporary headframe erected over the borehole for the purpose. The work was done by three men, and took relatively little time. The finished cost was found to be less than half the cost of a factory made borehole cable; while, as regards reliability, these home-made cables have been several years in use with no deterioration.

The underground equipment includes a large number of Sullivan chain coal cutters, both of the longwall and pillar and stall type. They are operated by 440-volt, 25-cycle induction motors and are started by means of a delta star controller.

Pumping outfits are of various sizes, all electrically driven. A typical example is an all-bronze pump, turbine type, 7-stage, pumping against an 800-ft head, the discharge pipe rising vertically through a large borehole to the surface; this pump is driven by a 150-h.p. motor, 2,200 volts, 1,500 r.p.m. Another, a 9-stage pump operating against an 1,100-ft head and discharging through a vertical borehole, is driven by a 200-h.p. motor. Some of the dip pumps are portable, mounted on trucks.

Underground hoists also are of various sizes, typically single drum, clutch operated, double reduction geared, and operated by a 440-volt slip ring motor with heavy steel case, gas tight and explosion proof. These motors are very heavy and extremely rugged. The controller is oil immersed to eliminate sparking.

Before closing, reference may be made to the Turnbull safety car, a mechanical device which has proved a great boon. It is used at the tail end of the trips being hauled out of the mine, instead of that abortion, the trailing bar. The safety car absolutely prevents runaway trips; for, the moment its wheels start to revolve in the opposite direction to that of hoisting, a pawl is tripped, releasing a heavy bar, which drops between the rails, locking the trip, which can only drop back down the incline a sufficient distance to set the locking bar. For lowering a trip, the pawl is set by a lever, so that the locking bar cannot be released. This device has rendered the hoisting of men much safer than heretofore. It may be classed, for slope hoisting, as the equivalent of the safety dogs for shaft hoisting.

Edison safety lamps are used exclusively in these mines.

## CONCLUSION.

It may be thought that the equipment at these mines is unnecessarily elaborate. This question resolves itself into one of comparison. Thus, the equipment is admittedly more elaborate than that of the average British Columbian coal mine but less so than is common in mines in Great Britain. The question as to whether such an installation is justified in British Columbian mines is best answered by comparing operating costs before and after.

As already mentioned, more than ten per cent of the output of the mines was, with the old plant, consumed in the production of an inadequate amount of power. No coal whatever is now consumed at the mines, except for railway locomotives.

The old plant included more than 30 boilers, fired by 42 stokers with an average payroll of $2,000 per month. The total cost of raising steam alone for these mines, allowing $2.00 per ton for the coal used, amounted to nearly $10,000 per

month — this does not include cost of maintenance and supplies.

As compared with this, the present average operating and maintenance cost for power amounts to less than $2,000 per month. In the figures given, both for the old and the new plant, no account has been taken of interest charges on the investment; nor, in the case of the new plant, has allowance been made for the revenue derived from the sale of electrical power to outside parties.

The power plant accounts are kept separately from those of the mines, which are charged a flat rate of one cent per kilowatt hour. The peak load factor is very high, amounting to over 85%. The output of the power-house is more than one million kilo-watt hours per month, and the charge of one cent per kilo-watt to the mines is admittedly high for this output. This, however, is merely a matter of book-keeping, and the power-house operation shows a large profit.

There are unquestionably many fully developed mining properties in the province which are still operating under great difficulties and disadvantages, due to inefficient and inadequate plants, entailing high operating costs and restricted output. In the writer's opinion, many of these might very profitably follow the example of Canadian Collieries and modernize their plants. One reason why such action is not taken is probably because mining men in general are so absorbed in the problems of developing and operating their mine that they cannot give the necessary consideration to the power requirements, despite the fact that the high cost and inadequate supply of power often means the difference between operating at a profit and at a loss. The logical remedy under such conditions is to place power plant matters in the hands of a competent power plant engineer.

The usual objection to the installation of adequate power plants at coal mines, is that the first cost is high. If, however, a coal mine is worth operating at all, it is certainly worth while to equip it with an adequate and efficient power plant. Given such a plant, the savings in cost of operation may be safely calculated to pay a handsome interest on the investment and at the same time output will be increased and improved.

# COAL WASHING ON VANCOUVER ISLAND.

By M. W. Garman.

Annual General Meeting, British Columbian Division, February, 1922.

In some coal-fields, the coal is so free from deleterious material that, except for dry screening preparation, nothing further is needed to produce a good fuel; but on Vancouver Island, this is not so. In earlier days, the chief product of a mine was lump coal, with perhaps egg or nut, all prepared by dry screening; the remainder was a wasted product. But to-day the value of small sizes is being recognized, so that what was once a waste product has been converted by washing into a saleable fuel, to the advantage of both the producer and the consumer.

In the following notes on coal washing on Vancouver Island, the seams of the Nanaimo field are the ones under consideration. In this field all the seams, except the Newcastle, make a large amount of fine coal, so that there is usually from 40 per cent to 85 per cent of screenings of less than 2-in. mesh, which goes into the washery. The Douglas seam, especially, often gives as high as 85 per cent of such screenings. It will be seen at once how large is the amount of fine coal that must be washed in order to work these seams profitably. Fines are the great source of loss in all coal washing, and this is nowhere truer than in the case of Vancouver Island coal.

It may seem paradoxical to say that coal washing should begin in the mine. Yet this is so, for several reasons.

Where a coal, like that of Vancouver Island, is friable, great care should be exercised by the underground management to see that all unnecessary breakage is eliminated, by careful use of powder in shooting down the coal, and by careful handling in loading and transportation. All impurities, such as rock, bone, and dirt, should be extracted as far as possible,

and where the purity of the coal at any working place is in doubt, a sample should, if practicable, be taken and analyzed. If it is found to be too high in ash, the place should be stopped; or, if it must be driven for haulage or ventilation, or for any other reason, the material should be gobbed — that is, left in the mine — and not sent out with the other coal. Unfortunately many regard a washery as a place where 100 per cent extraction can be made from any class of material handled. In reality the higher the ash content in the coal, the lower the possible recovery will be, the washery constituting only a part of the preparation plant necessary to produce a marketable product.

In the Nanaimo field, there are in use three general types of washers, namely, jigs of various types, as Lührig, New Century, Stewart; cone washers, as the Jeffry-Robinson, which is a combination of a hydraulic classifier and a centrifugal machine; and tables, as the Massco and Diester-Overstrom types.

Before taking up the system of washing in use at the mine where the writer is employed, a few general remarks concerning the nature of the coal and its impurities are desirable. The coal itself may be roughly divided into three classes with respect to its hardness: (1) a hard, dull grey coal; (2) a medium hard, lustrous coal; and (3) a dark, dull, soft variety. It must be understood that these divisions are not hard and fast ones—there are intermediate varieties connecting them.

The range of both specific gravity and ash content is very wide. The former ranges from 1.24 to 1.60, and the latter from 4.00 per cent to 26.00 per cent; but there is no apparent relation between the ash content and the specific gravity. The range of the latter shows that some coal will be lost in washing, because there is not enough difference between the specific gravity of the coal and the rock; also, the shape of the material has some bearing on the loss, flat, flaky pieces of rock having a tendency to go over with the coal. The refuse coming from the mine with the coal is chiefly rock, bone coal, shale, and clay, of varying specific gravities and ash contents. The following table shows the variable qualities of the coal and

the refuse, but is by no means a complete one. The arrangement in each case is according to increasing ash content.

| | Coal | | Refuse | |
|---|---|---|---|---|
| | Sp. Gr. | Ash % | Sp. Gr. | Ash % |
| No. 1 | 1.30 | 3.86 | 1.76 | 42.22 |
| No. 2 | 1.25 | 3.90 | 1.57 | 44.37 |
| No. 3 | 1.32 | 7.12 | 1.91 | 57.13 |
| No. 4 | 1.28 | 7.50 | 1.75 | 62.25 |
| No. 5 | 1.24 | 7.63 | 2.00 | 68.50 |
| No. 6 | 1.60 | 8.22 | 1.77 | 75.34 |
| No. 7 | 1.30 | 9.00 | 2.25 | 84.55 |
| No. 8 | 1.31 | 15.57 | 2.60 | 88.38 |
| No. 9 | 1.38 | 17.55 | | |
| No. 10 | 1.43 | 26.25 | | |

The process of washing at Cassidy is as follows: Mechanical feeders take the raw feed from the coal bin to two Stewart jigs of capacity forty tons per hour each. These jigs are of the double compartment type. In the first compartment the heavy rock and heavy bone are removed, and in the second the lighter bone and any rock which may have passed over from the first compartment. The raw feed consists of screenings from the lump coal screen of minus 2-in. mesh. Some screen analyses follow:

| (1) | |
|---|---|
| + 2-in. mesh | 0.0% |
| + 1-in. mesh | 14.3% |
| + $\frac{5}{16}$-in. mesh | 22.6% |
| + $\frac{1}{4}$-in. mesh | 4.4% |
| + $\frac{1}{8}$-in. mesh | 15.9% |
| + 20 mesh | 19.9% |
| + 40 mesh | 9.7% |
| + 60 mesh | 4.6% |
| − 60 mesh | 8.6% |

(2)

| | |
|---|---|
| Nut + 1-in. mesh | 10.9 to 18.05% ash |
| Pea + $\frac{5}{16}$-in. mesh | 28.6 to 17.30% ash |
| Slack − $\frac{5}{16}$-in. m. | 60.5 to 26.67% ash |

These analyses give some idea of the large amount of fines in the raw feed, and they show very clearly that great care ought to be exercised in mining, in order that these fines may be reduced to a minimum; as it is, much good fine coal is lost in washing.

As to whether sizing should precede or follow washing, this will depend on the particular plant and mine concerned; though, generally speaking, the writer believes that, where the coal is of a soft, friable nature, sizing after washing is best,

as it means a cheaper initial outlay, and the breakage can be taken care of by one sizing.

The jig bed at Cassidy is carried deep enough to prevent too great a loss of fines—12 to 14 inches in the first compartment, and 10 to 12 inches in the second. The material that goes into the hutch is chiefly fine rock and clay of a minus $\frac{3}{16}$-in. mesh. Occasionally, if the bed becomes thin for any reason, some fine coal of this size is lost into the hutch. In the jigs, one great trouble arises from the fact that the specific gravity of much of the impurities is not far removed from that of the coal; also the impurities are in large part of a flat, thin, flaky character, so that the buoyant effect of the water is greater than that of the mass of the particles. Consequently a good separation is not effected, many of the flat particles being washed over with the coal. There is another source of trouble, namely, that the carbonaceous clay, which is associated with the coal, forms a slimy, pasty mass with the jig water, and is difficult to get rid of. The coal becomes coated with a film of this mud, which gives it a bad appearance when dry and tends to increase its ash content. Spraying this coal has a beneficial effect.

From the jigs, the coal goes to a revolving screen where the washed nut and pea are separated from the slack. The pea goes into the washed coal bin along with the washed slack, while the nut is taken to the washed nut bin or loaded directly into railway cars.

The washed slack from the jigs, after being screened out from the nut and pea, is run into the sludge tank, from where it is passed by an elevator and conveyor to mechanical feeders, which feed it to the tables. A screen analysis of the table feed is as follows:

| | | |
|---|---|---|
| + $\frac{5}{16}$-in. | mesh = | 0.0% |
| + $\frac{1}{4}$-in. | " = | 2.6% |
| + $\frac{1}{8}$-in. | " = | 34.2% |
| + 20 | " = | 30.0% |
| + 40 | " = | 15.9% |
| + 60 | " = | 6.7% |
| − 60 | " = | 10.6% |

The tables used are the Diester-Overstrom, each handling from four to six tons per hour, with a recovery of from 88 to 93 per cent.

A screen analysis of the washed slack (A), and of refuse (B), from the tables shows the following:

| | (A) | (B) |
|---|---|---|
| + $\frac{1}{16}$-in. mesh .... | 0.0% | 0.0% |
| + $\frac{1}{4}$-in. " .... | 5.2% | ..... |
| + $\frac{1}{8}$-in. " .... | 27.6% | 23.4% |
| + 20 " .... | 46.0% | 29.0% |
| + 40 " .... | 14.6% | 20.4% |
| + 60 " .... | 3.7% | 13.0% |
| − 60 " .... | 2.9% | 14.2% |

The use of tables, with mesh openings of minus 5-16 inch and smaller, is most advantageous for the washing of fine coal; but a preliminary washing in the jigs is beneficial on the class of coal passing through the washery in this field, especially where carbonaceous shale and clay occur in such quantities, as quite a percentage of it is eliminated through the hutch, and, by carrying a good jig bed, little fine coal of good quality is lost.

No hard and fast rules can be made regarding table washing of coal, especially coal in this field. Various adjustments will be necessary before a suitable combination will give the desired results. These adjustments are chiefly for the following: (1) speed, or number of strokes per minute; (2) length of stroke; (3) horizontal inclination or elevation; (4) transverse inclination; (5) amount of feed; (6) regularity of quality of feed; (7) amount of feed water; and (8) amount of dressing water. These all have their effect on the washing of the coal. The mechanical feeding mechanism in use at this washery has been worked out on the plant to suit the prevailing conditions. It gives a regular, steady feed, such as is very essential in table washing in order to effect a good separation. The impurities to be eliminated are about the same as in the jigs, viz., rock, bone, clay, and shale; and here again, owing to the fact that both the coal and the impurities have a wide, and to some extent overlapping range in specific gravity, trouble is often incurred by coal being lost in the refuse, and by refuse going over with the coal. Especially is this true if the tables are crowded or overloaded, and if there is an extra quantity of fines with the feed — more particularly if the fines contain quantities of carbonaceous shale and clay. The writer has found that, generally speaking, from four to five tons per hour is a sufficient feed, consistent with good washing of this class of coal.

The usual indication of good washing is clean washed coal, and a high-ash rock. This, however, does not quite hold true here, for it has been found that a high-ash rock usually gives a high-ash washed coal; the reason being that some of the lighter refuse, such as shale and light bone, has floated over the rifling with the coal, while only the heavy — *i.e.*, high-ash — bone and rock go out with the refuse discharge. Test samples must be taken at frequent intervals, and daily assays for ash must be made, in order to determine the loss of coal and to work as economically as possible.

From the tables, the washed coal goes to a de-watering conveyor and from there to the washed coal elevator and into the washed coal bin. The fines from the upper and lower decks of this de-watering conveyor (which serves both as a conveyor of raw feed to the tables and also as a conveyor of washed coal away from them), are passed to two other tables where they are washed. The top deck fines are unwashed fines, while from the bottom deck the fines are those washed over with the coarser slack. Screen analyses of the feed, washed coal, and refuse from these tables are given below:

| | Feed | Washed Coal | Refuse |
|---|---|---|---|
| + $\frac{1}{16}$-in. mesh | 0.0% | 0.0% | 0.0% |
| + 20 " | 23.5% | 26.4% | 20.7% |
| + 40 " | 35.7% | 40.1% | 19.0% |
| + 60 " | 23.6% | 20.0% | 19.0% |
| − 60 " | 17.2% | 13.5% | 41.3% |

These tables can handle from six to seven tons per hour each, the feed averaging 21 per cent ash, the washed coal 14 per cent, and the refuse 45 per cent, a reduction of 33 per cent of the original ash content. The greater capacity of these tables, as compared with the others, is attributable to a better classified feed. A recent test gave the following results, and, as will be seen, the recovery was 94.6 per cent:

| | |
|---|---|
| Washed slack | 13.44 tons per hour |
| Refuse | .76 tons per hour |
| Total | 14.20 tons per hour |

The washed coal from these tables is conducted over a de-watering screen, and thence to the washed coal elevator. The wash water for the jigs is used over and over again, but the water for the tables comes from a Dorr thickener, which

clarifies the overflow wash water from the jigs, sludge tank, and elevating conveyor.

The water flowing into the thickener carries usually 6 per cent of solids with an average ash content of 35 per cent; the clarified water leaves the thickener with 0.15 per cent solids, a clarification of 99.9 per cent.

A screen analysis of the thickener discharge is as follows:

| | | | | |
|---|---|---|---|---|
| + 20 mesh | .... 0.0 % | .... 0.0 % | ash | |
| + 40 " | .... 0.25% | .... 0.0 % | " | (combined with+60 mesh) |
| + 60 " | .... 3.00% | .... 14.22% | " | |
| + 80 " | .... 7.00% | .... 16.97% | " | |
| + 100 " | .... 5.00% | .... 18.75% | " | |
| + 200 " | .... 35.00% | .... 25.60% | " | |
| − 200 " | .... 49.75% | .... 36.20% | " | |

giving an average ash of 29.40 per cent. The dilution of the discharge is 44.3 per cent solids, which amounts to about 100 tons per 24 hours. The inflow water is passed into an experimental oil flotation cell. Regarding coal flotation, the experimental results thus far are satisfactory, but some further work is necessary in order to complete details for good economic working results.

The following details of float and sink tests are given so that the results of washing may be seen at a glance. From them, also, the difficulties of washing Vancouver Island coal will be appreciated.

In the table below, the raw feed to the jigs is in all cases minus 2-in. mesh material, and the solutions in the several tests have the specific gravity indicated.

| | | | | |
|---|---|---|---|---|
| Solution 1.5 Sp. gr. — | Float | 77.5% ..... | Ash | 12.58% |
| | Sink | 22.5% ..... | Ash | 49.21% |
| Solution 1.6 Sp. gr. — | Float | 87.5% ..... | Ash | 15.61% |
| | Sink | 12.5% ..... | Ash | 56.50% |
| Solution 1.7 Sp. gr. — | Float | 88.0% ..... | Ash | 18.10% |
| | Sink | 12.0% ..... | Ash | 66.00% |
| Solution 1.3 Sp. gr. — | Float | 38.5% ..... | Ash | 7.21% |
| | Sink | 61.5% ..... | Ash | 29.38% |
| Solution 1.4 Sp. gr. — | Float | 71.5% ..... | Ash | 10.67% |
| | Sink | 28.5% ..... | Ash | 42.69% |

The average ash of the sample used for this test was 20.66 per cent.

**Raw feed to jigs:**

| | | | | |
|---|---|---|---|---|
| Sample 22.80% Ash — | Float | 56.0% ..... | Ash | 9.38% |
| | Sink | 44.0% ..... | Ash | 43.43% |

**Washed coal from jigs:**

| | | | | | |
|---|---|---|---|---|---|
| Sample 14.30% Ash — | Float | 77.67% | ..... | Ash | 10.16% |
| | Sink | 22.33% | ..... | Ash | 26.68% |

**Refuse from jigs:**

| | | | | | |
|---|---|---|---|---|---|
| Sample 67.05 % Ash — | Float | 2.5 % | ..... | Ash | 16.00% |
| | Sink | 97.5 % | ..... | Ash | 68.16% |

**Float and sink tests on table products:**

| | Float | Ash | Sink | Ash |
|---|---|---|---|---|
| Feed to tables..... | 59.00% | 10.00% | 41.00% | 40.00% |
| Washed slack..... | 71.00% | 9.50% | 29.00% | 31.32% |
| Rock..... | 4.00% | 14.65% | 96.00% | 46.60% |

**Float and sink tests on tables washing fines from de-watering conveyor:**

| | | | | | |
|---|---|---|---|---|---|
| Feed to tables..... | Float | 69.0% | ..... | 12.50% | Ash |
| | Sink | 31.0% | ..... | 41.49% | Ash |
| Washed Slack..... | Float | 70.0% | ..... | 11.10% | Ash |
| | Sink | 30.0% | ..... | 35.20% | Ash |
| Refuse..... | Float | 1.5% | ..... | 20.67% | Ash |
| | Sink | 98.5% | ..... | 53.73% | Ash |

The writer hopes by these notes to convey some idea of the complexity of Vancouver Island coal, and of the difficulties encountered in preparing it for the consumer's use. It may not be out of place here to call attention to the lack of knowledge on the part of consumers, and especially the domestic consumer, of coal, regarding the value of the smaller sizes as now prepared by the Island washeries, as compared with lump coal, which is now in chief demand for domestic use. Lump coal commands the highest price; but after buying it the consumer immediately breaks it into smaller sizes, often making considerable slack or fine coal. Washed No. 1 and No. 2 nut, and pea coal, are eminently suited for domestic consumption, being quite equal in calorific value to lump, with the added advantage of being decidedly cheaper.

# NOTES ON COAL FLOTATION.

By P. E. Peterson.

Annual General Meeting, British Columbian Division, February, 1922.

The writer has carried on some experimental work to test the suitability of the flotation process for the cleaning of Vancouver Island coals. Samples and washery products obtained from the Douglas seam were mainly employed for these tests. For the laboratory experiments, a Janney flotation test machine was used. Larger tests were made with a Peterson cell, eight feet six inches in diameter, having an approximate capacity of fifty tons per twenty-four hours. The results of the work show that, on the material tested, good recoveries of clean coal can be made by the use of the process.

The flotation process does not depend on the specific gravity of the substance in order to make a separation (although in floating coal advantage is taken of the differences in specific gravities), but rather on the formation, in a watery pulp, of an oily froth that has an affinity for the coal or carboniferous substances. A flotation machine is an apparatus that by violent agitation or aeration produces a suitable froth from a proper mixture of oil, water, and pulp, and by an arrangement of mechanical skimmers or launders removes the froth from the surface of the watery pulp and thus effects a separation of the coal or other substance.

*Suitable Oils.*—Almost any oil will give good results. Of all the oils tried, only gasoline failed to give a froth. In general, the wood oils, such as wood creosote, pine oils, tars, and turpentine, give good voluminous froths, while mineral oils, such as crude petroleum, coal tar, coal tar creosote, and kerosene, produce heavily loaded compact froths. Organic oils, such as castor oil, yield froths similar to those obtained with the mineral oils.

The selection of an oil depends a great deal on its cost, on the size of the coal particles to be floated, and on the type of flotation machine used. Some machines, as for example those of the mechanically agitated, skimmer type, are perhaps

best suited for operation with a heavily loaded compact froth; machines of the subaerated type, such as the Callow cell, operate best with a voluminous froth. Froths of any character required can be obtained by the mixing of wood and mineral oils in various proportions. The selection of an oil mixture depends greatly on the nature of the coal to be treated. This is especially so with Vancouver Island coals, in which, apparently, the ash is inherently different in the several coal fields. The separation of low-ash products requires the application of selective flotation methods, such as a limited amount of oil and repeated re-cleaning of concentrates. The best results were obtained when using an oil mixture that produced a voluminous froth.

*Quantity of Oil Required.*—The amount of oil needed varies somewhat with dilution of the pulp.

A dilute pulp containing 95 per cent water will require about one-third more oil to float a given quantity of coal than a pulp containing 75 per cent water. The actual weight used also varies with the kind of oil, but in general ten pounds of oil is all that will be required to float a ton of coal, and in some cases two pounds of oil per ton of coal floated will be enough.

*Size of Coal Floated.*—All sizes of coal and carboniferous material passing through a ten-mesh screen are effectively lifted by the froths.

In copper mineral flotation, sizes coarser than forty-mesh will not very readily be lifted by the froths. This large difference in size of the coal particles, and of the metalliferous minerals, that will float, is accounted for by their difference in specific gravity, that of coal being 1.4 as against about 4.3 of chalcopyrite.

The fineness to which it is necessary to grind the material for flotation depends entirely on the nature of the coal to be treated. Material low in coal, such as washery refuse, requires fine grinding for the greatest recoveries. To ensure a good recovery of clean coal from waste products containing 50 per cent ash or over, it is necessary to grind the material sufficiently fine to pass through a hundred-mesh screen.

*Other Factors (Dilution, etc.).*—Cleaner coals are recovered when working with dilute pulps. The ordinary dilution of 75 per cent water gives a dirty coal, but if from 90 to 98 per cent water is used clean coals are readily obtained. This is no doubt due to the clay, which forms a large proportion

of the rock in coal, and which, when ground with water, makes a colloid that is very slow in settling, and is thus carried in the float along with water in the froth.

Either fresh or salt water may be used without affecting the result.

Economy in the use of oils is obtained by adding them to the pulp in a mixing cell ahead of the frothing cells. In fact, it is difficult to obtain uniform results in pneumatic cells unless the oils have been thoroughly mixed with the pulp prior to flotation.

*Typical Results obtained.*—(In all the tests cold water, having a temperature ranging from 7 deg. C to 10 deg. C., was used)

**Flotation Test on Jig Rock. — Test No. 1.**

| Product | Recovery % Total Wt. | % Ash |
|---|---|---|
| Heads | 100.00 | 50.00 |
| Concentrate No. 1 | 44.60 | 29.00 |
| Concentrate No. 2 | 29.20 | 26.00 |
| Concentrate No. 3 | 23.20 | 22.40 |
| Concentrate No. 4 | 19.20 | 19.88 |
| Concentrate No. 5 | 14.15 | 14.34 |
| Concentrate No. 6 | 11.90 | 13.16 |

The above tests show the variation in recovery with the varying ash content of the coal made. In making a coal having 13.16 per cent ash, a recovery of 11.90 per cent of it is obtained, whereas when making a coal having 26 per cent ash, there is a recovery of 29.2 per cent.

**Flotation Test on Mine Screenings. — Test No. 2.**

| Product | Recovery % Total Wt. | % Ash |
|---|---|---|
| Heads | 100.00 | 24.15 |
| Concentrate No. 1 | 96.60 | 22.50 |
| Concentrate No. 2 | 90.50 | 20.80 |
| Concentrate No. 3 | 87.10 | 19.40 |
| Concentrate No. 4 | 86.40 | 19.07 |
| Concentrate No. 5 | 83.00 | 18.20 |
| Concentrate No. 6 | 78.50 | 17.00 |
| Concentrate No. 7 | 74.80 | 16.00 |
| Concentrate No. 8 | 72.10 | 15.50 |
| Concentrate No. 9 | 69.40 | 14.70 |
| Concentrate No. 10 | 66.20 | 14.20 |
| Concentrate No. 11 | 54.70 | 11.80 |

If a 17 per cent ash content in coal is desired, the foregoing tabulation shows a recovery of 78.50 per cent can be obtained.

The following table is a summary of the results obtained when treating materials of varying ash content:

Per Cent Weight of Coal Recovered in Concentrate.

| Material Tested | % Ash | % Total W't. 11% Ash | % Total W't. 12% Ash | % Total W't. 13% Ash | % Total W't. 14% Ash | % Total W't. 15% Ash | % Total W't. 16% Ash |
|---|---|---|---|---|---|---|---|
| Jig Rock..... | 50.00 | 10.4 | 12.0 | 13.0 | 14.1 | 15.0 | 16.3 |
| Jig Rock..... | 40.00 | 22.5 | 24.5 | 27.0 | 29.5 | 32.0 | 34.5 |
| Bone Coal.... | 32.90 | 37.0 | 40.0 | 43.1 | 47.1 | 51.1 | 55.0 |
| Fines in sludge tank overflow.... | 29.35 | 43.0 | 47.0 | 51.0 | 55.0 | 60.2 | 65.1 |
| Mine screenings... | 24.20 | 49.5 | 54.5 | 60.2 | 65.0 | 71.1 | 75.0 |

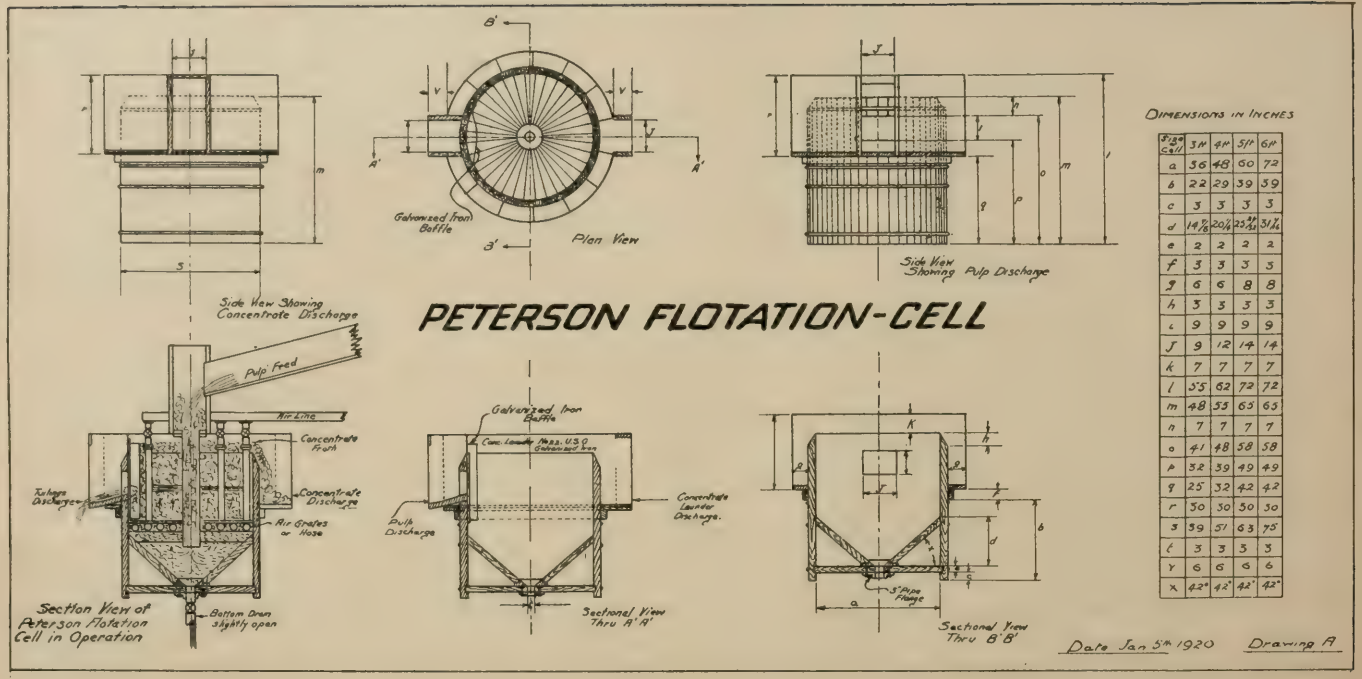
The accompanying diagram shows the type of pneumatic flotation cell used in the large scale experiments.

The principal features of this cell are its simplicity and the low cost and ease of installation. The cell requires practically no attention. The water level can be changed at will and remains at any set point until changed. The spigot discharge from the bottom of the cell effectively prevents banking. Per unit of cross-sectional area, the capacity is as great as in any other cell.

*Application of Flotation to Coal Washing.*—Flotation is best suited for the recovery of coal from fine material. The cost of recovering coal by the flotation process is greater than the cost of recovering it by jigs or tables; and naturally the latter method, when applicable, will continue to be employed in preference to the use of flotation. Coal washing tables are an effective means of cleaning coal coarser than sixty-mesh, and it is only with sizes finer than this that flotation will be useful.

The flotation process will not revolutionize the present practice of coal washing, but it will serve as a useful adjunct in recovering the very fine coal from slimes that are produced in the present practice. In the saving of coal from washery waste where it is necessary to grind the material, flotation in conjunction with coal tables will be the principal means of effecting the recovery.

*Disposal of Flotation Concentrates.*—On account of the extreme fineness of flotation coal, it is necessary to de-water it by means of Dorr thickeners and filters. The de-watered product from the filters is then mixed with other sizes of coal and sold to the users of pulverized coal.




DIMENSIONS IN INCHES

| Size of Cell | 3 Ft | 4 Ft | 5 Ft | 6 Ft |
|---|---|---|---|---|
| a | 36 | 48 | 60 | 72 |
| b | 22 | 29 | 39 | 39 |
| c | 3 | 3 | 3 | 3 |
| d | 14 7/8 | 20 1/4 | 25 29/32 | 31 1/2 |
| e | 2 | 2 | 2 | 2 |
| f | 3 | 3 | 3 | 3 |
| g | 6 | 6 | 8 | 8 |
| h | 3 | 3 | 3 | 3 |
| i | 9 | 9 | 9 | 9 |
| j | 9 | 12 | 14 | 14 |
| k | 7 | 7 | 7 | 7 |
| l | 55 | 62 | 72 | 72 |
| m | 48 | 53 | 65 | 65 |
| n | 7 | 7 | 7 | 7 |
| o | 41 | 48 | 58 | 58 |
| p | 32 | 39 | 49 | 49 |
| q | 25 | 32 | 42 | 42 |
| r | 30 | 30 | 30 | 30 |
| s | 39 | 51 | 63 | 75 |
| t | 3 | 3 | 3 | 3 |
| Y | 6 | 6 | 6 | 6 |
| X | 42° | 42° | 42° | 42° |

Date Jan 5th 1920 Drawing A

# THE UTILIZATION OF PULVERIZED COAL.

By J. Thomas Dovey and George N. Calkins.

Annual General Meeting, British Columbian Division, Vancouver, February, 1922.

Powdered coal has been in use for a period of over 25 years in metallurgical furnaces and in cement kilns, but it was not until 1913 that a plant was constructed by the Dominion Coal Company, of New Waterton, Nova Scotia, for the use of powdered fuel in stationary boilers. This was the first plant on the North American continent to use powdered fuel for this class of boiler. Operators in the eastern part of the United States followed this development more closely than did we of the West, and though there are a number of cement and other companies in Canada and in the state of Washington — including the B.C. Cement Company, at Todds Inlet, B.C.; the Superior Portland Cement Company, at Concrete, Washington; the Olympic Cement Company, at Bellingham; the Lehigh Portland Cement Company, at Metaline Falls; the International Portland Cement Company, at Irving, Washington; the Bunker Hill & Sullivan Mining Company, in the Cœur d'Alene district; the Tacoma Smelter; and others — using pulverized coal, there are, as yet, comparatively few plants using this fuel in stationary boilers.

The largest installations that are operating with powdered fuel in stationary boiler plants in this vicinity are the B.C. Sugar Refining Company, at Vancouver; the Puget Sound Power & Light Company, in Seattle, at its Western Avenue plant (4,100 H.P.); and the plants of the Pacific Coast Coal Company, at Newcastle, Washington (1,500 H.P.), and at Briquetville, Washington (200 H.P.). In addition, there are a number of similar plants of smaller size.

## Low Grade Coals Adaptable.

The low-grade bituminous and lignite coals of Washington and vicinity are well adapted for use in powdered form. Results have shown that, in this method of burning, low-grade coals are quite as satisfactory as the higher grades. A striking example of this is seen at the Western Avenue plant of the Puget Sound Power & Light Company in Seattle, which,

since the spring of 1918, has been operating with powdered coal obtained from the sludge dump at its mine at Renton. This washery sludge had been accumulating for many years and was regarded as waste, its small size (about 50% passing through a 10-mesh screen) rendering it unfit for use on any form of mechanical stoker. Its composition, on a dry basis, is approximately as follows: volatile matter 37%, fixed carbon 38%, ash 25%, and sulphur 1%. The material carries about 25% moisture, and it has an average heat value of 7,300 B.T.U. It is thus much lower in heat value and higher in moisture and ash than many of the Washington lignites. Yet the use of this sludge has at all times proved satisfactory and economical, and boilers are regularly operated on continuous loads of 150 to 300% of rated capacity, with a marked reduction in boiler maintenance.

Similar satisfactory results have been obtained by the Pacific Coast Coal Company in their plants at Black Diamond and Coal Creek, where coal containing approximately 16% moisture and 20% ash has been used to economic advantage in powdered form. Tests show the Black Diamond plant to be saving 4 cents per 1,000 lb. of evaporation, or a net saving of $51.21 per day over hand firing, besides reducing the labour from 18 to 15 men per 24-hour shift. At the Coal Creek plant, the evaporation per pound of coal fired rose from 4.49 to 7.40 lb. with the introduction of powdered coal, the boiler efficiency was increased 65%, and the consumption of coal per hour, with the development of the same horse power, was reduced from 7.1 to 4.3 lb.

### CENTRAL DISTRIBUTING SYSTEM.

The Pacific Coast Coal Company is the only organization in this part of the country that is manufacturing and distributing pulverized fuel commercially at the present time, or which is making use of a central distributing plant with the necessary facilities for placing the product within reach of the consumer. The plant of this company is located almost a mile north of Renton, on the east side of Lake Washington, and has connection with two railways in addition to the transportation facilities afforded by the lake. Briquets also are manufactured at this plant, as the operation of preparing the coal for briquets and for powdered fuel is the same up to the point of grinding.

Briefly stated, the plant consists of two 15-ton (hourly capacity) rotary driers of the Ruggles type, and two five-ton Raymond pulverizers, with the necessary conveying machinery. The raw coal is fed by chain conveyor from the receiving bunker to the driers, where it is heated to a temperature of 150° to 200° F., according to the moisture content. It leaves the drier with an average moisture content of from 2% to 5%, depending on the character of the coal. Discharging from the driers, a system of elevators carries it to the pulverizing mill, where it is ground to a fineness of 85% passing a 200-mesh screen and 95% passing a 100-mesh. Thence, by means of spiral conveyors, it is carried to a point over a specially constructed railroad car, into which it is discharged through large canvas tubes. The bunker for the reception of the product at Seattle is of 85-ton capacity, equipped with bucket elevator and screw conveyors handling about 10 tons per hour. It is equipped with canvas chutes for delivery to specially designed motor trucks, which distribute the product directly to the consumer.

### OIL-BURNING EQUIPMENT CONVERTIBLE.

The adaptability of this class of fuel for use in plants already equipped for burning oil is one feature that should particularly recommend it to the fuel consumer. Such plants may be converted to the burning of powdered coal with very little change of equipment, since the burning of coal in powdered form more nearly approaches the burning of oil than that of any other form of fuel; and the change can be made the more readily where facilities exist for the delivery of powdered coal from a central pulverizing plant, as above described.

In this connection, reference may be made to the plant of the Standard Furniture Company, in Seattle. Formerly operated with oil, this has now been replaced by powdered coal, and the recently installed equipment is perhaps the most complete and up-to-date of its kind in the country. The receiving bin, with three compartments, giving a total storage capacity of 65 tons, is placed under the alley, and absorbs no additional space from the original plant area. Each of the compartments has independent drives and feeds. The boiler plant, used both for heating and lighting the building, consists of two 150 H.P. Babcock and Wilcox boilers and is

equipped with recording instruments of the latest and most improved type, including Venturi feed water meters, feed water temperature indicator, stack temperature thermometer, and carbon dioxide and draft recorder.

This installation is equipped with automatic controls dependent upon the boiler pressure. It is not an uncommon occurrence to see a pressure card running from eight o'clock in the morning until five o'clock in the afternoon with a variation of only three pounds — this without any attention of the fireman. The fact that the load at this plant fluctuates considerably, due to the operation of elevators as well as the handling of the heat and light loads, makes this an all the more noteworthy performance.

Besides the undoubted economy of powdered fuel, it has the added advantages of quick starting, continuous action, and absolute control and regulation, which means uniform heat and rapid and complete combustion — the latter, by eliminating the smoke nuisance, being an especial desideratum in a city plant.

The Pacific coast has kept abreast of developments in the use of pulverized coal and at the present time there are a number of powdered coal installations in this region, not a few of which are within the heart of the business district of Seattle. These have been operating so successfully, and with such absence of smoke and soot, that the average citizen has not known of their existence, despite the fact that they are ranged side by side with oil burning installations.

A number of mechanical devices applicable to small installations have been put on the market from time to time, but, to the writer's knowledge, none of these have proved entirely successful in their operation. While no doubt many of them have points of merit, many mechanical details remain to be worked out before the small installations for apartment houses, domestic furnaces, etc., can hope to be commercially successful.

## DELIVERY BY TRUCK.

The method of delivery by motor trucks from the bunker to the consumer is simple, clean, and efficient. The trucks, specially designed for the handling of powdered coal, are filled at the bunker by gravity and are discharged, by the same

method, through openings equipped with tight fitting ratchet gates, directly into the bunker of the consumer. A small pipe at the top of the consumer's receiving bin is connected with a similar pipe running to the top of the truck tank. This facilitates dumping and makes the operation entirely dustless, as the air which the coal displaces in the bin enters the emptying tank.

In filling the truck, the body is tilted at an angle of 45 deg., making; at that time, a tight connection with the spout from the bunker. In its delivery at the sidewalk level, the powdered coal flows directly into the bin beneath, without waste, dirt, or the use of extra power of any kind. The capacity of the truck is five tons, which amount is discharged through the two openings in approximately twenty minutes. The bins for the reception of the powdered coal at the various plants have capacities ranging from 6 to 65 tons and are built of steel or concrete — the latter has been found preferable in most of the installations.

While there is considerable variation of the density of powdered coal, 40 lb. per cubic foot is the general average. The material flows readily, the angle of repose being somewhat less than 60 deg. Storage bins and conveying machinery are made as air tight as possible in order to prevent circulation or renewal of enclosed air, as well as to keep the coal from absorbing moisture: for as long as no moisture is allowed to come in contact with the coal there is no possibility of spontaneous heating.

## SIMPLICITY OF EQUIPMENT.

The extreme simplicity of the equipment leaves little leeway for description. It consists simply of the necessary pipe connections, and worm feeds from bunker, blower, and burners. A motor-driven worm feed, directly under the storage bin, draws the coal from the bin; delivers it through the transmission pipe on top of an air current generated by the blower and varying in pressure from 1½ to 3 lb.; and carries it to the furnace with the proper amount of admixed air.

The blowers used for the air admixture and for carrying the coal to the furnace may be constant or variable pressure types depending on the location of the fuel supply.

Many different types of burner have been designed for the discharge of the coal into the fire box. All of them possess certain points of merit.

## PRINCIPLES OF USE.

The principles underlying the use of powdered coal were so clearly set forth in an article in the engineering supplement of the London *Times* of February 22nd, 1918, that they are herewith quoted:

"Combustion of fuel consists of two distinct processes: (1) the distillation of the volatile hydro-carbons and their union with the oxygen of the air; and (2) the combustion of the remaining fixed carbon of the fuel. For the first process it is necessary to have a high furnace temperature so as to quickly drive off the hydro-carbons, and also to have an adequate supply of air thoroughly mixed with these gases in order to provide the necessary oxygen for combustion. Sufficient space must also be available in the furnace to allow the combustion process to be completed before the gases come into contact with the cooling surfaces. For the second process it is necessary that the air supply be so arranged that every particle of fixed carbon is supplied with oxygen for combustion, and that the temperature is sufficient for the process to be completed. It is probably true that for both these processes the most important item is the supplying of the correct volume of air and the thorough mixing of this air with every particle of the hydro-carbons and the fixed carbon."

In this connection, however, I might state that, in furnaces where the volume is restricted to a very low combustion area, compared to the horsepower of the boiler or to the quantity of coal burned per hour, steam is used at the end of the burner where the coal is discharged into the boiler. This has a tendency to localize the flame, or, in other words, it gives a Bunsen effect to the flame, raising the temperature of the furnace to a very high degree — so much so that, in some grades of coal, the ash is clinkered to such an extent that it is very difficult to remove it. To continue the quotation from *The Times:*

"When ordinary fuel is being burnt, even with the latest automatic stoking and air regulating appliances, it is exceedingly difficult to adjust the air supply correctly; and conse-

quently the volume of air necessary, even in first-class boiler installations, is at least twice the amount which is theoretically necessary for perfect combustion. The loss thus caused is probably the greatest that takes place in the use of fuel in ordinary boiler installations.

"In pulverized fuel plants the fuel is injected into the furnace in a very finely divided state and is mixed in its passage to the furnace with the correct volume of air. This mixture is injected into a chamber of suitable shape and volume, which is maintained at a high temperature. As each particle of the fuel passes into the chamber, the gases are immediately driven off, and for these, and the remaining particles of carbon, oxygen is available for immediate combustion; consequently the process is so quick, and the conditions so good that a high temperature is maintained and combustion considerably improved.

### GREAT AREA OF SURFACE EXPOSED.

"It is interesting to consider the surfaces which are exposed with the fuel as ordinarily used and with the same weight of (powdered) coal. In the case of a piece of coal one inch cube, the surface exposed is 6 square inches. If, now, this cubic inch of coal is pulverized so that each particle is a cube, the length of whose side is one-hundredth of an inch, the surface exposed by the whole of these small particles will be 600 square inches. Further, each one of these particles will be surrounded with a supply of oxygen, and, given a sufficient temperature and room, the completion of the combustion process is much better assured than with the piece of coal one inch cube."

Some authorities have maintained that very fine grinding of the coal is not necessary, and that they have obtained just as good results with coarser particles. No doubt in such cases there has been sufficient furnace volume and travel of the coal to ensure complete combustion before reaching the tubes of the boiler. However, it is the writer's opinion that coarse coal would not burn economically in short or compact furnaces. The conclusion seems self-evident that, the finer the particles, the more instantaneous is their ignition.

When it is considered that the generally accepted standard of fineness requires 85% of the powdered coal to pass through

a 100-mesh screen which contains 10,000 holes to the square inch, and 75% to pass through a 200-mesh screen of 40,000 holes to the square inch, it will be readily seen that the surface exposed in one cubic inch of the material is almost inconceivably large, and, as each infinitesimal particle is surrounded by oxygen, the combustion, at sufficient temperature, must be complete.

## LARGE ECONOMIES POSSIBLE.

The economies that have been effected by proper air regulation and admixture, such as is possible with powdered coal, have ranged from 20% to 50%. In this connection Mr. Leonard C. Harvey, a noted English fuel engineer, in his report on Pulverized Coal Systems in America, prepared for the Fuel Research Board of Great Britain, makes the following statement: "Of the 513,500,000 tons of coal used in America in 1914 for all purposes, of which, say, 205,400,000 tons were used in metallurgical and steel industries and for railway locomotives, some 41,000,000 tons would have been saved if pulverized coal had been used exclusively for these purposes."

## PRESENT STATUS OF THE INDUSTRY.

The consumption of powdered coal in the United States at present is about 15,000,000 tons annually. Of this amount, approximately 10,000,000 tons are used in the cement industry, in rotary kilns. In this industry, powdered coal, on account of temperature and output being under complete control, has practically supplanted all other types of fuel. Of the 110 cement plants operating in the year 1916, 91 were using powdered coal, 17 oil, and two gas or oil and gas. It should be added that the plants using oil are located in States where oil is the regional fuel — such as Washington, Oregon, California, Arizona, and Texas. The output of these plants aggregated about 92,000,000 barrels of cement in that year, of which over 80,000,000 had been manufactured in plants operating with powdered fuel.

The metallurgical industries consume about 2,000,000 tons of powdered coal annually in various types of furnace,

such as open hearth, heating, puddling, soaking pits, continuous heating, re-heating, annealing, forging, etc., and some of the largest and most up-to-date plants in the United States are equipped for its use.

The particular advantages of powdered coal in the metal trades are the decrease in oxidization and the fact that, when the metal is brought up to welding temperature, as in skelping, busheling, and tube welding, it is softer to work. Soaking pits and forging furnaces, if fired with powdered coal, produce a more easily worked billett, the heat soaking through the metal more uniformly than in furnaces fired with oil or gas.

In the discussion regarding the Anaconda Copper Company's installation at the Washoe reduction works, Professor J. W. Richards, in the *Journal of the American Institute of Mining Engineers*, states: "Comparing coal dust (powdered coal) firing with grate firing, the difference is, that in burning coal on a grate, not more than two-thirds or three-quarters of the heat generated by the coal by combustion goes into the furnace. There is a loss of 25 to 33 per cent of the calorific power of coal by radiation from the sides. By powdered coal firing this is obviated." It was further stated in the discussion that, on the introduction of powdered coal firing at Anaconda, the cost of smelting was reduced 50 cents per ton. Based on the figures given in this discussion, the comparison between grate firing and coal dust firing for one million tons of copper ore (the United States production of copper in 1916 having been approximately 875,000 tons) would be: grate firing reverberatory furnaces, 258,000 coal tons; powdered coal fired furnaces, 140,000 coal tons; or a saving of over 100,000 tons of coal per million tons of charge smelted.

Approximately 1,500,000 tons of powdered coal are used annually in connection with the zinc, gold, silver, fertilizer, potash, feldspar, shale, and dolomite industries, and in miscellaneous steam generation.

In the year 1920, one firm of manufacturers of powdered coal equipment in the United States had orders placed with them representing an annual pulverizing capacity of 1,200,000 tons.

With the advent of improved types of mills for grinding, together with more efficient methods of transporting powdered coal, more ways will be found for its use, and an increasingly larger output can be looked forward to from year to year.

## DISCUSSION.

PROF. JOSEPH DANIELS: One of the most important points in the question of the use of powdered coal concerns itself with the utilization of material now regarded as waste, owing to its being too small or too high in ash to be efficiently used in ordinary processes of combustion. A large percentage of fine sizes and much bony coal now being mined and prepared in the regular course of operations must be discarded or sold at a loss, thereby throwing an additional burden on the operator or on the purchaser of the coarser sizes and better grades marketed. The question is one of economics as much as technology. The extended applications of the use of pulverized coal in the metallurgical and other industries indicate that earlier objections to its use are being overcome, and, if the increasing demand for pulverized coal will absorb the waste products referred to, part of the losses now falling on the industry will be eliminated. In addition, areas of lower rank coal, which are now non-competitive, will become available, leaving the higher rank coals for utilization in select services. It does not appear, however, that the use of coal in powdered form is going to make it possible to use coals indiscriminately. The questions of ash content, nature of the ash with respect to chemical composition and fluxing properties, and thermal and reduction values of the coal, are of great importance to the metallurgical and industrial user, and must be considered in the light of availability, efficiency, and relative cost, as compared with the corresponding factors in the use of oil or unpulverized coal.

# THE TRANSACTIONS

OF THE

# MINING SOCIETY OF NOVA SCOTIA

1922

# PAST AND PRESENT METHODS OF WORKING No. 2 MINE, SPRINGHILL, N.S.

By J. C. Nicholson.

Annual Meeting, Mining Society of Nova Scotia, Sydney, May, 1922.

No. 2 mine of the Cumberland Railway & Coal Company, at Springhill, N.S., is on the No. 2 seam, which is the lowest of the three principal seams in that district, the other two being respectively 100 ft. and 280 ft. above it. This slope was opened in 1873, and with the exception of stoppages during several strikes — with which the people of this province are more or less familiar — it has been in continuous operation since that time.

The slope is driven on the pitch of the seam, which runs in a northerly direction. The average dip from the surface to the 3,300-ft. level is 30 degrees, with a maximum at one place of 38 degrees; from there to the bottom, which has now reached a total length of 6,900 ft. along the slope, the dip is uniformly at 20 degrees. At the 2,800-ft. level a down-throw fault was encountered which necessitated driving a stone tunnel for 100 ft. This is the point of maximum grade.

The coal, which has an average thickness of 9 ft., is of excellent quality and has the reputation of being the best steam and domestic coal in Nova Scotia. The output at the present time is 1,500 tons per day.

The mine is entered by three slopes — haulage slope, downcast air slope, and pipe slope. At, roughly, 600 ft. intervals, levels are broken off, and these are designated by their distance from the surface measured along the slope, as the 1,300-ft. level, the 1,900-ft. level, and so on.

The original method of working, with the exception of slight deviations due to some local condition which usually occurs in every seam, was to divide the coal into pillars in the first working. As the levels advanced, headings spaced 700 ft. apart were driven on the full pitch of the seam, and the coal was brought down these places on sheets, by gravity where the grade was sufficient, otherwise by boys called "shovers down". When the heading had reached the level above, sheets

were taken up and replaced by track, for a back balance, using a cage. Then bords were broken off on both sides of the balance and driven 350 ft., or until they holed through on the workings coming from the next balance. This procedure was continued until the level had reached the determined boundary, and the whole lift, or block, of coal between the two levels had been completely divided into bords and pillars. Then, beginning at the inside, the pillars were attacked and brought back toward the slope.

The first pillars were 30 ft. thick, with cross-cuts every fifty feet. Their dimensions were increased at different times until, at the 3,300-ft. level, pillars were 40 by 70 ft., and the width of bords was reduced from 12 ft. to 10 ft.

This method worked satisfactorily on the upper lifts; but as the working got deeper and the cover increased, it was found that, before all the pillars could be brought back, much of the first working had badly caved, thus necessitating cleaning up of falls and in many cases the driving of entirely new bords, which were driven adjacent to, and on the high side of, the old ones. This operation is called locally "ribbing in". In fact, on one occasion, after suspension of operations for two years due to labour troubles and a series of delays, the bords and balances at the 3,300-ft. level (where the cover had increased to 1,600 ft.) were found to be so badly fallen that, before the pillar work could be resumed, the whole lift had to be again divided up into bords and pillars.

This experience led to a material change in the method of working. On the next lift levels were shortened, the size of the pillars was increased to 60 by 90 ft., and no tight work above the entries — with the exception of an occasional heading to shorten the air-course — was begun until the entries had nearly reached the boundary and were ready to come back with the pillars.

But even this change did not have the desired effect. The cover had now reached a thickness of 2,000 ft. The floor began to heave badly, with the result that, even where the pillar work was begun immediately on completion of the bords, it was found impossible to remove them without much "ribbing in". Trouble from the same cause was experienced on the balances, the heaving of the pavement throwing the track out of alignment and reducing the height, so that there was not

sufficient head room for the cages. Moreover, the seam developed a tendency to bumps — these increased in frequency and severity to such an extent as to cause the management considerable anxiety. It was evident that, if the mine was to be worked at a profit, some radical change must be made in the method of working. Larger pillars could hardly be considered, as already in cross-cuts and pillar work — on account of the flat pitch — the coal had to be shovelled two or three times before it was loaded into the mine cars, and this was a source of constant complaint with the miners.

After much consideration and some experimenting, the following method was adopted:

Three entries, as before; haulage level, water level, and counter level are driven to the boundary. Headings in pairs are laid off every 500 ft., but are driven only to the counter level, with the exception of an occasional one to shorten the air course. Between the headings, and from the water level, or minebord, to the haulage level, cross-cuts are driven at 100 ft. centres, thus making the entry pillars 90 by 90 ft. When the entries have nearly reached the boundary, the headings are continued to the level above, beginning at the innermost one and starting the others successively until four are in operation. Coal from the heading is lowered by jig wheels until the headings break through, when the wheel is replaced by a drum. Three landings, forty-five feet apart, are then cut at the top; from these, bords are driven on both sides, making the pillar between them 35 ft. The bords are driven 250 ft., or until they hole through to the working from the next inclines, when the pillars are immediately brought back. The face of the bords and the pillars are kept at least 40 ft. ahead of the bord immediately below, and not more than three landings in one incline are working at one time. As one bord is finished a new one is begun, until the coal is brought back to the level.

The system of haulage on the inclines is known as the cut rope system. One of the ropes has two detachable sections, equal in length to the distances between the landings. These are readily detached or connected as the case may require. When running from the top bord, both are detached and laid aside; when running from the others, one or both sections of

the rope are connected, according as they are running from the middle or lower landing. The landings are crossed by means of bridge rails. Perhaps this will be shown more clearly by the accompanying illustration (Figure 1).

At this point it may be of interest to make a few remarks of a more or less theoretical nature. Now, at first glance, a 35-ft. pillar, under a cover of 2,500 ft., may appear contrary to the first principles of mining; but such is not quite the case.

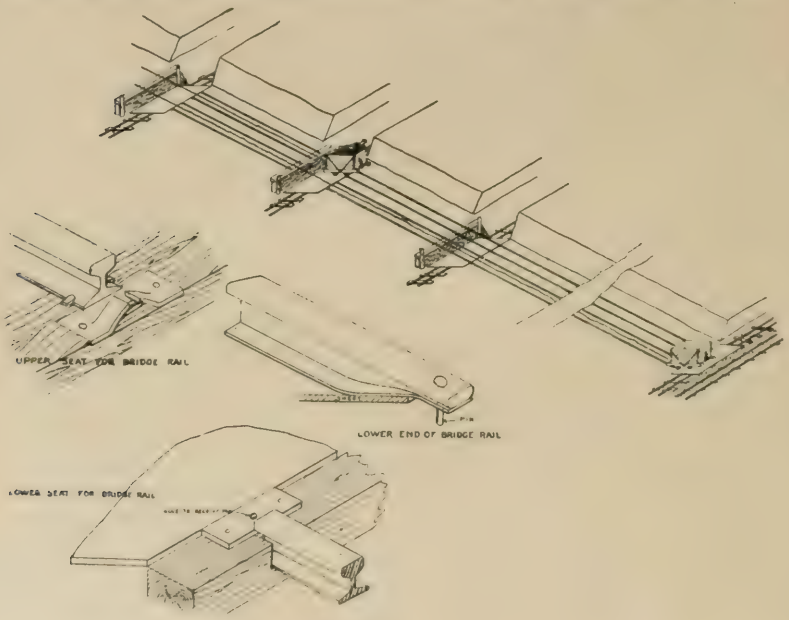
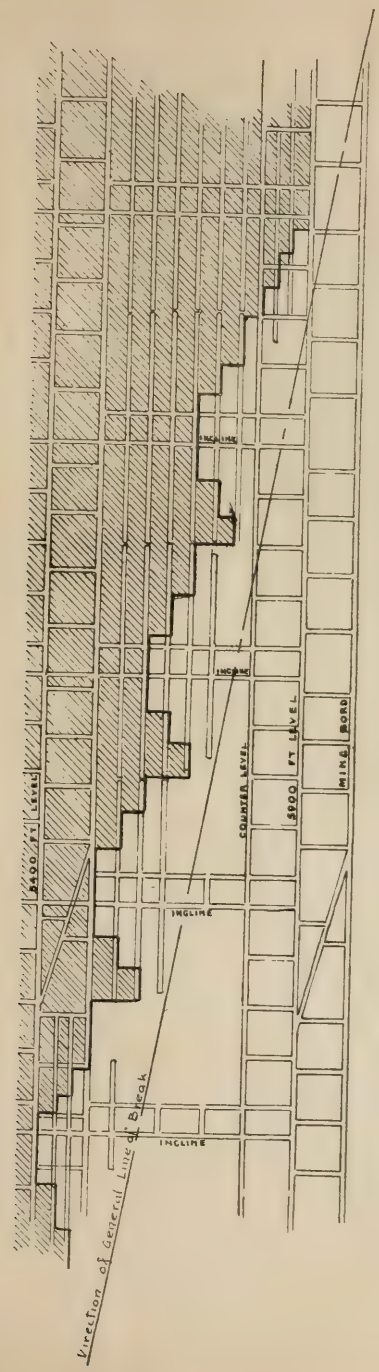



**Figure 1**

It is one of the fundamental principles of mining that the size of the pillars must increase with the depth; but another principle, equally important but sometimes overlooked, is that the size of pillars depends on the time they are to be left standing. It is to the observance of this latter principle that the present system owes its success. The pillars are removed so quickly that the bords have not time to cave.

Another feature which led to the adoption of so small a pillar was the problem of "bumps". It was early observed that these were more severe, and occurred more frequently, in driving the bords with a large pillar than with a small one.




IDEALIZED PLAN OF WORKINGS - No 2 MINE SPRINGHILL N.S.

Figure 2a

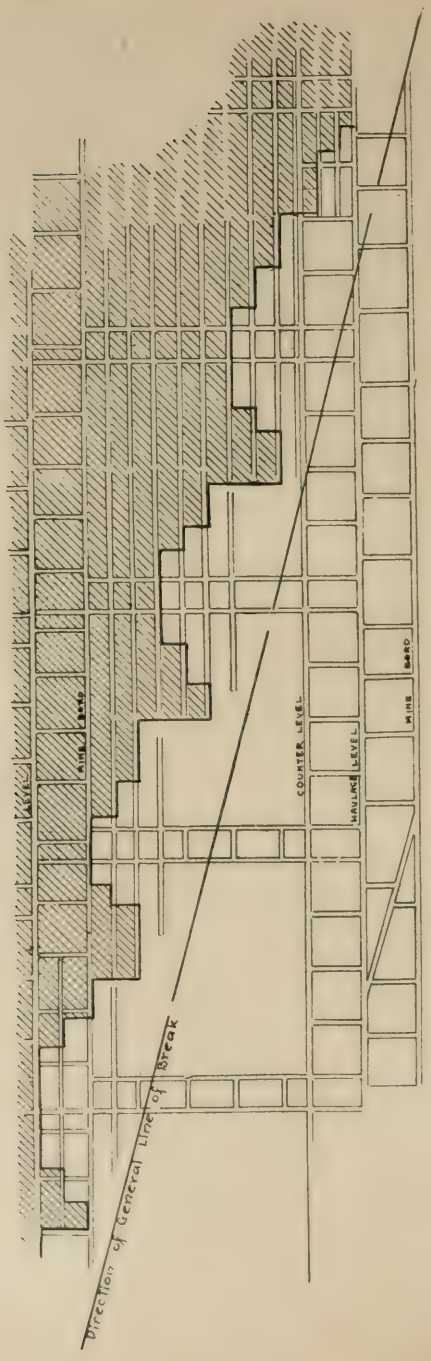

IDEALIZED PLAN OF WORKINGS - Nº2 MINE SPRINGHILL N.S.
Direction of General Line of Strike
COUNTER LEVEL
HAULAGE LEVEL
MAIN SHAFT
MAIN SHAFT

Figure 2b

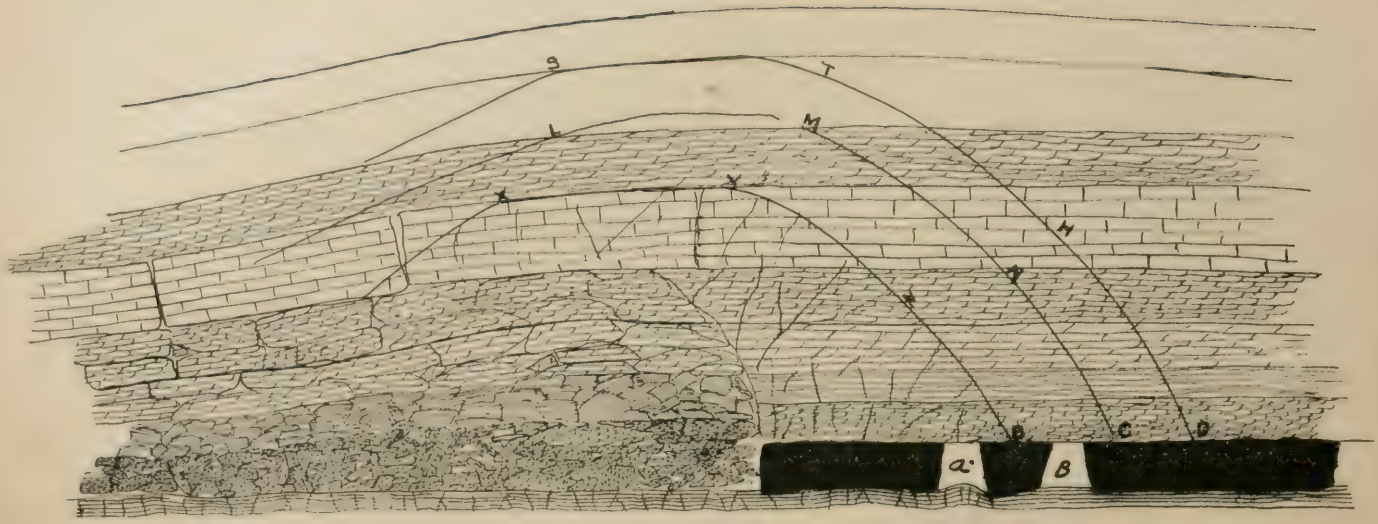
As a matter of fact, the first pillars were 45-ft., but the bumps were equally frequent and more severe. (See figures 2A and 2B for plan of workings.)

Before proceeding, it might be advisable to explain just what is meant by "bumps". The term "bumps," I understand, is a more or less local term, used to describe a sudden jarring of the mine due, apparently, to a yielding of some great force in the strata. The bumps occur unexpectedly and without giving any warning, which distinguishes them from other disturbances of a similar nature, such as "crushes" and "creep". Apart from the noise, bumps consist chiefly in upheaval of the pavement and dislodgment of large quantities of coal. They are not generally accompanied by falls of roof, except where the roof has been previously broken, in which case there have been falls of considerable extent.

These bumps are accompanied by some very wonderful phenomena. For instance, in one of the most severe, the jar was felt in the town as far back as three-quarters of a mile behind the outcrop, and it was so great as to rattle the dishes on shelves in houses: yet, at the same time, directly over the workings it could scarcely be detected. At another time a bump occurred in a bord where a pack, or crib, 7 ft. high and made of 4-ft. sticks, had been built on the high side — after the bump this crib was found tight against the low side, with not a stick displaced.

For the purpose of this paper, bumps in this mine may be classified as *local* and *general*. The former, which are confined more or less to one working place, and invariably in the tight work, consist chiefly of pavement upheaval and displacement of coal. The "general" bumps extend over a considerable area, including the pillars, and are usually accompanied by falls of roof. These, as might be expected, have been the most destructive of property, but the local bumps have caused the greater number of accidents. I merely make this statement as a matter of record — it must not be construed to mean that the general bumps are inherently less dangerous, for some of the escapes in these have been little short of miraculous.

Now to return to the relation between small pillars and bumps. When a portion of a coal seam is removed, the remaining coal is subject to abnormal pressure, due to the extra weight




B Point of maximum pressure if roof breaks to arch X.Y.Z
C " " " " " " " L.M.N
D " " " " " " " S.T.H.

Figure 3

brought on by the arching of the strata over the waste. This abnormal pressure would be at a maximum at some point, whence it gradually diminishes until it reaches the normal. It is evident that this maximum is at some point beyond the edge of the solid coal, say at "B", Figure 3, and that it is continually moving forward. This movement might be brought about in two ways: in the first place, by the coal yielding and sinking into the pavement; and in the second, through the point of maximum pressure being at the base of the arch, and the latter becoming wider as it increases in height.

Now suppose a bord driven in the solid at, or just ahead of, the point of maximum pressure ("B" in Figure 3) where the coal and pavement have not already yielded. As the coal is removed, the pavement is free to heave; but being strong, it does not do so at once — perhaps not until the bord is driven some distance; but then, as it is very brittle, it yields suddenly, causing a bump. Had the bord been driven between "B" and the waste, the pavement and coal, having already yielded, would not give way suddenly, but slowly, and there would be no bump.

Now while this supposition may be sufficient to explain small bumps, we shall have to seek further to account for the larger ones.

So far, we have assumed the point of maximum pressure to be moving gradually, and the pavement and coal to be yielding suddenly. I think it reasonable to assume that the point of maximum pressure will change erratically. Accepting our previous assumptions, this would be the case if the falls took place in large masses, say by strata; then, when a stratum falls, the pressure would advance instantly, causing the coal and pavement, which are already near the breaking point, to give way with a bump — the magnitude of the disturbance depending on the thickness and extent of the stratum which falls. From this it will be noted that the prime cause of the disturbance is not necessarily a sudden increase in the pressure, but rather a sudden shifting of its point of application, thus producing a new set of forces to which the strata must adapt themselves. This explanation of the cause of bumps is but a

theory, yet it seems to me it presents fewer objections than any other explanation so far offered.

As to the prevention of these troubles, what has been said already will show our method of dealing with the local bumps. The general bumps, fortunately, have not been of very frequent occurrence, and thus they do not offer so much on which to base an opinion. It has been noticed, however, that where they have occurred, the general line of break has had a decided bulge into the coal. If we assume a thick stratum, strong enough to support itself and the roof above it until the bulge has covered a considerable area, but which then gives way suddenly, I think our explanation of these phenomena not unreasonable. So, by keeping the line of break straight and leaving no pillars in the waste, we hope to avoid, or at least to decrease, their destructive effect.

This is a necessarily brief description of the present system of working in this mine, and of the conditions which led to its adoption. We do not claim perfection for it, since, like every other method, it has some undesirable features. That there are only six working places in each incline makes the output per incline much lower than if it were practicable to use balances with cages and, consequently, for a given output the work is not concentrated, as is considered good modern practice.

Also, for the same reason, the general line of break makes too small an angle with the strike. While this may not be a disadvantage in the actual working of the coal, it evidently brings too much pressure on the entries, thus making the cost of upkeep rather high.

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## DISCUSSION.

MR. ALEX. MCEACHERN: I have listened to Mr. Nicholson's paper, and I may say frankly that when I first heard that they advocated small pillars in Springhill I felt there was something wrong. Every writer on the theory and practice of mining, from the first time that I knew anything about it until the present day, has said that the more weight you put upon the base of a pillar the stronger you must make it. I am not criticizing the Springhill practice. I had an opportunity of

visiting the mine last year, and I may say that I was agreeably surprised to find that, so far as my humble judgment goes, they were absolutely right. But they had not told us the full story. To my mind their success is due to two things. Firstly, to narrowing up the bridge to about 30 or 35 feet. I think Mr. Nicholson will admit that if he doubled that they would have serious trouble. The second reason is that they retreated the top pillars before they completed the lower. Each pillar had to take its own work from the thrust, due to the pitch, with the result that there was a uniform pressure put on each pillar. Whoever is responsible deserves credit, but they do not deserve credit for hiding certain things. They do not deserve credit for leading us to believe they had worked miracles. I have not anything to say against the system, excepting that I think it is perhaps the only one that would carry small pillars under the existing conditions in Springhill.

MR. J. C. NICHOLSON: Mr. McEachern is quite right in his conclusions about the work at Springhill. As the paper explained, we tried two systems before we arrived at the one adopted. The top bord is driven nearly 40 ft. before the second, and so on. There is only 35 ft. of space at any time, which is probably the only reason we can carry so small a pillar. I have seen them smaller than 35 ft. I think our system is about the only one we could have at the present time. We have no desire to change it.

MR. W. HERD: Mr. Nicholson has given us a very interesting paper. I am sorry I did not hear the first part. It is very rarely we hear of mines being troubled with bumps. The bumps at Springhill are caused by a certain physical condition which exists there. They have a very hard roof and a hard pavement, the roof being the harder of the two. Although the floor is so hard that it will not heave even at great depth, when conditions arise such that something must give, it is the weaker, which is the floor, that gives. It really bursts quickly, and this causes a bump. I think there is only one other place in the world where they have bumps to the same extent and through the same cause. At Springhill we may be at a critical depth and as we get deeper the weight of superincumbent strata may cause the floor to break oftener, thus gradually releasing the pressure. I will ask Mr. Nicholson

if he has ever known a bump to occur in Springhill where the floor was soft? In almost all cases the bumps have occurred where the floor was hard. It might be suggested, therefore, that, as a preventive, artificial means might be adopted to break the floor and so relieve the weight. This, I believe, was done in Fernie, where gas was said to be the cause of the bumps. The question of the small pillars is interesting. I was one of a certain committee which originally discussed reducing the size of pillars. It seemed opposed to all the known principles of mining, but practice has proved that it is the best safeguard yet tried to prevent bumps. Mr. Nicholson is to be congratulated on having given the Society a paper which is probably one of the most interesting that has ever been presented before it.

MR. J. C. NICHOLSON: When the bottom is soft we are not troubled with serious bumps. We have bored holes in the pavement, but they did not do much good. The bumps came from the weight.

MR. W. HERD: I would like to make one more remark, and that is that we have heard a lot about the weight throwing over, due to the high inclination. I believe that if we had the seam absolutely flat with a hard roof and a hard pavement, but the roof harder than the pavement, we would still get bumps.

MR. JOHN MOFFATT: I think Mr. Nicholson's paper lacks one thing. He did not go far enough back. Mr. Nicholson dealt with conditions as he finds them, and he referred only incidentally to the past methods of mining. I would not like to contradict a man like Mr. Nicholson. He says that there are gas bumps. I wish to state that, for the last three or four months, I have been sitting at the feet of Mr. McEachern and Mr. Herd, and I have been learning all I can. Mr. Herd, in telling me about bumps, referred me to the Crowsnest Pass. Bumps are largely due to rigid roof and to bad methods of mining. They are not so much due to the gas (and gas is found in very large quantities in the Crowsnest Pass) as to the bad methods of mining. It was necessary for them to adopt new methods of working there, so that to-day they are getting a much larger extraction percentage, showing that they are working along the same lines as in Springhill.

# MINE PUMPING PROBLEMS.

By Karl Marsh.

Annual Meeting, Mining Society of Nova Scotia, Sydney, May, 1922.

At the Dominion Coal Company's Cape Breton mines, about 5 tons of water are pumped to the surface for each ton of coal mined — the amount varies at different collieries from 0.5 to about 13.2 tons. Of the total power consumed at these mines, about one-third is used for mine pumping; about one-third of the electrical power is used for main mine pumps, and approximately one-third of the total compressor capacity for sinking and relay pumps. The labour and materials used in pumping these mines is a big item — but is less than half the cost of pumping power. Since the workings are, in most cases, advancing seawards with the dip, mine pumping is becoming a more and more serious matter, and some interesting problems are involved.

Although the working faces of most of the mines are below sea level, and many of them are in submarine areas, there is little if any infiltration of sea water. It is surface water principally that filters into the mine, and thus the "make" of water is to a certain degree seasonable, depending on rains and melting snows. To prevent this water entering the mine is not always an easy problem, but every effort has been made, by draining swamps and ponds, to provide good surface drainage to the sea. At some mines, unfortunately, the pillars have been drawn near the outcrop, so that it is now practically impossible to prevent the infiltration of surface water.

Every reasonable endeavour has also been made to impound the leakage into the mines at the minimum depth below the surface. Immense lodgments have been formed to catch the seepage from melting snows and heavy rains, thus protect-

ing the workings from rushes of water and permitting these reservoirs to be economically pumped at a slower rate than the maximum "make". To further minimize the considerable expense of mine pumping, air operated pumps are being replaced by electric motor driven pumps, large and highly efficient electric pumping units are being installed to replace several small and less efficient electric pumps, and furthermore, the cost of electric power is being reduced by improvements and economies at the power generating stations. It is the purpose of this paper to point out some of the problems met in installing and operating these main electric driven mine pumps, with the idea of promoting a discussion on such features as may be of interest to the members of this Society.

At points where current is taken into the mine, we are using the standard outdoor type of oil-cooled transformer substation, and find this construction satisfactory and considerably less expensive than the indoor type. The transformers are fully protected with choke coils, lightning arresters, expulsion type fuse blocks of the disconnecting switch type, and oil circuit breakers. A Watt hour meter is installed on the transformer secondaries. In some cases the secondary disconnecting switches are remotely controlled from the mine pump house, so that, in case of fire, the pump operator may cut off the current supply at the sub-station. It is proposed to arrange the secondary oil switch at the sub-station with a balanced relay to automatically open in case the mine cable or pump motor becomes grounded. The whole sub-station is surrounded by a high paling fence, surmounted with barbed wire.

Mine cables are of the 3-conductor type, lead covered and steel armoured. The armour is always galvanized and is always thoroughly grounded at the surface. A separate ground wire is invariably used in parallel with the cable armour. For bore hole cables, the self-supported type is quite successful for moderate depths, but below 500 ft. we prefer to support the bore hole conductor on a separate stranded galvanized wire suspension cable, to which the lead covered conductor is securely attached by tarred jute serving. A continuous and water-tight serving of tarred jute is usually specified for bore

hole cables, as any leakage of water through the bore hole casing has a tendency to destroy the steel armour, and, if the water is corrosive, to damage the lead covering also. To protect suspension cables against corrosive water, a coating of Stockholm tar is used.

Mine conductors are mechanically connected below ground; no soldered joints are used. The switch gear for mine pumps should be damp proof and dust proof and designed to eliminate sparking; no wood or fibre should be used in its construction. The wiring in mine pump-houses should preferably be lead covered and steel armoured, and that section between the motor and starter should be in conduit also. On account of its tendency to spark, a slip ring motor is objectionable for driving a pump in a gaseous mine. We prefer Squirrel Cage induction motors having open slot construction, moisture proof impregnated windings, and form wound coils. The motor frame and all switch gear frames are thoroughly grounded.

Bore holes for electric conductors and for water discharge are lined with a steel pipe casing, and cement grout or concrete is poured between the casing and the walls of the drilled hole. In deep, wet holes, it is difficult to make absolutely sure that the steel casing is surrounded with a permanently water-proof jacket, so that in the case of a wire hole, corrosive water may attack the steel pipe from the outside and thus find its way to the conductor cable and cause considerable damage. In water holes discharging corrosive water, the steel casing is quite certain to become partly dissolved, resulting in undue friction and perhaps leakage through the surrounding grout, so that some water may enter the rock strata and thus find its way back into the mine. The sinking and lining of these bore holes is a hazardous undertaking at best, and there is need for some improved method of making these bore holes water-tight. Possibly a cast-iron casing could be used to advantage, or perhaps some modification of the cementation process might be used to seal the strata more effectively than the ordinary grouting process. Bore holes into our submarine mines are sunk as near as possible to the shore, and the water from these holes sometimes discharges to the sea through a short tunnel driven in the cliff slightly above high tide.

The lodgment for a main mine pump is usually planned well in advance of the time when the pump will be required, and every endeavour is made to provide for the desired quantity of storage with a minimum of expense for packs and dams. It is desirable to have one or two scour pipes, with valves arranged at the lowest point in the lodgment, so that silt may be occasionally drained out if required. In our mines, the pump is usually arranged below the lodgment, because the maximum depth of water in the lodgment is greater than the suction lift of a pump; furthermore, the possibility of leaks in a long suction line under-ground makes it very desirable to have a positive head on the supply pipe to a pump. To prevent a pump suction from becoming clogged, it seems best to have it enter the lodgment well above the pavement level and to protect its end with a suitable wooden screen and a suitable trash rack. A 'basket' type strainer is always installed in the pump suction line, near the pump. It is considered good practice to arrange two independent suction pipes into the lodgment, so that either may be used in event of the other becoming clogged. Concrete dams are well reinforced with steel and substantially keyed into the pavement, roof, and ribs. Sometimes the dam is made in the form of a frustum of a pyramid, so that water pressure on the dam tends to tighten the joint in the keyways.

Flanged cast-iron piping, hot dipped in asphaltum or pitch varnish, is preferred for all mine pump piping, whether the water is good or bad. For pressures up to 750 lb. we prefer plain faced flanges with the finished portion of the flange slightly raised inside the bolt circle and faced with a rough cut so that the gasket cannot easily slip and blow out. For moderate pressures, rubber gaskets are good, but for high pressures fibre gaskets are preferred. There is a process of centrifugally casting iron pipe which is widely used for making bell and spigot pipe, but which has not been perfected for making flanged pipe. This process produces a dense and close-grained material somewhat stronger than sand cast iron, and the walls of the pipe are of uniform thickness and quite smooth. If flanged pipe were made by this process, it should find wide application in the mining industry for use in laying water and air lines.

Wood pipe is often used in small sizes for handling extremely corrosive water at moderate heads. No data is available on the durability of fibre pipe in Cape Breton mines, and it is suggested that this subject be investigated. Another fruitful subject for research would be the determination of the best type of paint or protective coating for use on underground steel pipe lines.

The successful operation of a mine pump depends to a considerable extent on the design of piping, and we have found it advisable to make these pipe lines as straight as possible, and especially to avoid sharp bends as well as air pockets and dips in the line where silt might collect. Mine water usually causes an incrustation on the inner surface of pipes, and this increases the frictional resistance to a remarkable degree. We therefore endeavour to design the pipes a trifle large, and seldom use velocities exceeding 7 ft. per sec.

Stop valves in pipe lines handling bad water deteriorate rapidly. For high heads we use all-iron valves with solid gates and monel metal stems; no brass or bronze valve parts are permitted. For moderate heads, cast-iron gate valves having beech-wood wedges and lignum-vitæ seats are often very successful. Gate valve bodies are frequently lined with silica cement or sheet lead and give splendid satisfaction. The wedges of these valves are usually solid cast-iron with hard babbitt face rings. A plug type of valve made of ferro-silicon has been successfully used for several months with bad water, and this type of valve construction is the best we have used. The Hadfield Company are making stainless steel castings, and they claim that this material is suitable for valves handling mine water.

For a permanent mine pump station, a substantial pump-room is a good investment. This room should be well ventilated with intake air, well drained, and properly illuminated; no combustible material should be used in its construction. We prefer masonry walls and floors, steel doors, and a roof designed to prevent moisture from dripping on pumping machinery. A hand crane of simple construction, arranged to lift all pump and motor parts, is essential for economical construction,

good pump operation, and quick pump repairs. Every pumphouse should be equipped with a chemical fire extinguisher and a box of sand.

In selecting a mine pump, the hydraulic conditions do not necessarily govern the type of pump best suited for the work. First consideration must be given to reliability — economical performance is next in importance. The character of the water to be pumped, whether corrosive or non-corrosive, clear or gritty, has an important bearing on the choice. Most permanent mine pumps operate against constant head and with a positive pressure on the suction, and these conditions favour the choice of centrifugal pumps. For gritty waters, however, centrifugal pumps may cause trouble, due to the abrasive action of the grit on the shaft and packing. In purchasing a centrifugal pump for efficient operation, the pump builders should be fully advised as to the details and arrangement of piping, the working heads, and variations in speed.

The European builders of centrifugal pumps have had much more experience with bad mine waters than have Canadian and United States manufacturers; and the Swiss, especially, have been very successful in building highly efficient and durable pumps for handling corrosive mine waters and very high heads.

For quantities up to 500 G.P.M. and heads up to 500 ft., the Dominion Coal Company make, in their shops, a 1,500 R.P.M. centrifugal pump that is fairly efficient and very successful in handling bad mine water. The cast-iron casing is lead lined, the impellers and diffusion rings are of hard babbitt, and the shaft is of monel metal.

For pumping extremely corrosive water to high heads, say 500 to 1,000 ft., we usually employ a plunger pump of the pot valve type, preferably triplex or quintuplex. These pumps are made of cast-iron and are lined with either lead, rubber, or silica cement. The plungers are usually cast-iron, lead or babbitt covered. Chilled cast-iron plungers are quite successful in some waters. Porcelain covered plungers resist corrosion and grit, but are structurally weak for high pressures. A glass covered plunger will be tested in service shortly. The correct

packing of plungers is of extreme importance; those too tightly packed cause undue friction, and those loosely packed leak badly and erode rapidly. U-shaped rubber packing rings have been developed which quite effectively prevent grit and water leaking past them to the softer hemp packing next to the gland. With this type of packing, the wear of plungers is said to be considerably less than with ordinary ring packing.

The high pressure plunger pumps are likely to leak at the joints, so that some water squirts out for a considerable distance; and in order to protect the pump motor, we prefer a rubber belt drive so arranged that a steel or masonry wall effectively prevents this water reaching the motor and switch gear.

In pumping corrosive water to high heads, the pump valves and valve seats offer a most difficult problem, as the destructive action of the bad water increases rapidly with the head and velocity. So-called acid resisting bronze has not been successful under these conditions. Lead and babbitt valve seats resist the corrosion very well, but lack the necessary hardness and strength. Frary metal, a hardened lead, is somewhat stronger and harder than ordinary lead or babbitt, and is being used for valve seats with a considerable degree of success. For very bad mine water, chilled cast-iron valve seats stand well for a week or two, but the iron soon becomes dissolved, leaving a carbon skeleton that is soft and weak. Hard rubber, bakelite, and glass, have all been used for valve seats with varying degrees of success. For high heads, bakelite seems superior to hard rubber as a valve facing.

In selecting anti-corrosive materials for mine pump construction, we find a laboratory analysis of the water very helpful. The relative amounts of sodium hydrate required to neutralize equal volumes of different mine waters, is a good indication of their relative corrosive properties. Most of our bad mine waters contain little free acid, but they do contain sulphates of aluminium and ferric-iron, which are very corrosive. Laboratory tests of metals are very useful in determining the selection of those best suited for use in pump construction, but these tests are by no means a sure criterion of the results to be expected in practice, because the varying conditions of

pressure and velocity inside a pump cannot well be duplicated in a laboratory. The selection of pump materials is best made by observing the action of the mine water on pump parts in actual service.

In some gold and diamond mines, the water is neutralized with caustic and filtered before pumping, but this process is unduly expensive for coal mines. Very beneficial results are noted, however, when oil or grease is injected into the pump suction, along with the corrosive water. This lubricant appears to stick to the inner surface of the pipe and pump, thus forming a protective coating which is of material assistance.

The United States Bureau of Mines is now conducting an investigation into the corrosive action of mine waters, and the subject is of such vital importance to the coal operators of this district that it is suggested that members of this Society co-operate with the Bureau insofar as possible.

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## DISCUSSION.

MR. W. HERD: Pumping water from mines requires much attention, and this is especially true in the case of our mines, owing to the corrosive nature of the water. Mr. Marsh raised one very interesting point, that of the material from which to construct valves. It is very important, and opens up a big field for research. I do not think we know of a metal for valves that will stand our corrosive water for long. We have tried a great many different metals, but they lasted only a limited time. Another point Mr. Marsh raised was the type of cable that should be used. I cannot quite agree with the use of the lead covered cable. I think that, in the past, a great many mines have had trouble with lead cables. On the question of bore holes, we have not had much trouble up to the present, because the bore holes have been comparatively shallow; but as we get deeper, it will be difficult to prevent

some of the water leaking back through the strata. Possibly in the future it will be necessary to sink, say, three bore holes of smaller diameter a few feet away from the main one and, at a very high pressure, inject cement so as to make the strata around the main bore hole thoroughly impervious.

MR. LIONEL FRIZZLE: What interests me most is the bore hole cables. It is, admittedly, a debatable subject, but my experience is that the lead covered steel armoured cable is the best for bore holes. The bitumen cable failed on one occasion.

MR. ALBERT DAWES: Mr. Marsh spoke about a type of pump successful for quantities up to a head of 500 feet. Above this, a plunger pump is preferred. Recently we have had a lot of discussion in regard to a pump to be installed on the North Side, and reports are now being made as to what the type should be. I must confess I feel as if I am getting old, for I still seem to favour the reciprocating pump for high heads. I do believe that, for pumping against enormous heads with economy of operation and reliability, the reciprocating pump still has the most points in its favour. I believe that, before a general type of pump is decided on, every point for and against either type should be most carefully considered. I learned something from Mr. Marsh's paper with regard to the different classes of metals being used for plungers. There is certainly a great field here for research, which I hope the Dominion Coal Company will undertake.

# EXPLOSIONS AND THEIR CAUSES

## WITH SUGGESTIONS FOR PRECAUTIONARY OR PREVENTIVE MEASURES TO BE OBSERVED.

By Alexander McEachern.

Annual Meeting, Mining Society of Nova Scotia, Sydney, May, 1922

Since the writer first entered the mining profession, some forty years ago, mine explosions have taken place at different times and in various countries. Mining men everywhere have sought the cause of these terrible and sudden catastrophies. Theories to account for them have been advanced from time to time, but, until comparatively recent years, the generally accepted view was that mine gas, or "fire damp", was mainly responsible. The gas might be ignited in various ways—as by the flame of a lamp; a flame from a blown out shot, or a windy shot; as the result of a shot that was overcharged, and which, having done its work, finished in the shape of projected flame; or through an adobe shot, or an electric spark—but however the ignition took place, the explosive gas was regarded, in nearly all cases, as the *cause* of the explosion.

About twenty-seven years ago, while employed as underground manager at Old Bridgeport colliery, I remember on one occasion, after a number of shots had been fired, seeing flame pass out of a room and travel past the room-end while we stood in the headway; and I felt at the time that there was some other agent than explosive gas at work. The mine had then an output of 800 tons per day, and the total quantity of air or ventilation returning to the furnace was only 18,000 cu.ft. per minute. There was no explosive gas present in this part of the mine at the time, and one would naturally suppose that the flame would die out because of the lack of oxygen, and also on account of the presence of a certain percentage of carbon dioxide.

In the No. 2 Mine, Glace Bay, there were four occasions during 1902 on which men were badly burned by flaming shots.

In one case, three men were very seriously burned; three mine officials had examined this place before the shots were fired, and the miner also had examined the place for gas, but none was found. After this the use of black powder had to be discontinued, and compressed powder was resorted to in order to prevent flaming. In some of the cases of flaming shots, even the coal caught fire, and was extinguished with difficulty. The writer finally became convinced that gases liberated from the coal, or generated by the explosive (chiefly carbon monoxide), acted as an agent to set off the explosion, and that these were stimulated by the coal dust, as there was no explosive gas present in the places where this flaming occurred.

Although many mining men in this country doubted that coal dust, in the absence of explosive gas, played any part in these explosions, watering of the whole mine, by means of pipe lines, was then applied for the first time in Eastern Canada. In addition to this, the dust was gathered up and loaded away.

In respect to precautionary measures against coal dust explosions, we were perhaps as far advanced as many of the European countries. The first record of the influence of coal dust in mine explosions is contained in the report of John Buddle on the Wallsend explosion in 1803. He states that the mine workings were dusty, and some of the survivors were burned by sparks from ignited dust. In 1844 Faraday and Lyell drew attention to the effect of dust in extending the explosion in the Haswell colliery in that year. Speaking at the Royal Institution, Faraday said: "The ignition and explosion of the fire damp raised and kindled the coal dust, which is always pervading the passages, and these effects must in a moment have made the part of the mine which was the scene of the calamity, glow like a furnace."

M. Vital, in 1875, proved that fine coal dust would increase the intensity of a fire damp explosion. Wm. Galloway came to the same conclusion in the same year. But notwithstanding all this, in 1882, M. Mallard and M. Le Chatelier, members of the French Fire Damp Commission, recording the results of their investigations, said that coal dust was not a serious danger, and that no colliery explosion of any magnitude could be traced to it.

Thus, up to 1891, the large scale experiments of Galloway, Abel, the Chesterfield Institute, and the more important ones of Hall and the Austrian Commission, were inconclusive. In the final report of the Austrian Commission, it was stated that their experiments confirmed the results of their previous investigations, but showed that the dangers from coal dust were much greater than had hitherto been admitted. They concluded that black powder and similar explosives were dangerous, and should be prohibited in gaseous mines. They also showed that the explosibility of coal dust depended more upon its physical than upon its chemical composition. They classified the various kinds of dust as (a) explosive, (b) slightly explosive, (c) inflammable, and (d) safe.

A Royal Commission, to investigate the question of coal dust explosions in mines, was appointed in England in February, 1891, and began hearing evidence in the following month. Its first report, without conclusions, was published in July of the same year. In 1892 its sittings were adjourned pending further experiments by Mr. Hall, which were conducted at intervals in that year and in 1893. The Commission made its second report in 1894, giving the additional evidence and their final conclusions, which they summarized in the following words: (1) The danger of mine explosions in which gas exists is largely increased by the presence of coal dust. (2) A gas explosion in a fiery mine may be intensified and carried on indefinitely by coal dust raised by the explosion itself. (3) Coal dust alone, without the presence of explosive gas, may cause a dangerous explosion, if ignited by a blown out shot, or a violent inflammation; but to produce such results, however, conditions must be exceptional, and are only likely to be produced upon rare occasions.

A few years later a second Royal Commission was appointed, and began hearing evidence in June, 1906. The second report of this Commission appeared in 1909, and the coal dust question was discussed, together with other kindred matters. Witnesses, who included those best qualified by scientific knowledge and practical experience to deal with the subject, expressed opinions which were often widely divergent, but which generally agreed that coal dust was liable to explode, with or without the presence of fire damp.

After the Courrieres explosion in France in the year 1906, a testing station was established at Lievin to study the relative explosibility of the different coal dusts; and, as had been proved at other European testing stations, it was found that there was no difficulty in causing explosions of coal dust without the presence of fire damp, by discharging black powder or dynamite from a cannon at the end of the gallery. Although there was no official report attributing the Courrieres explosion to coal dust, it was the general opinion of the English investigators that this was the cause.

In the United States, the greatest coal producing country in the world, the coal dust factor attracted scant attention until 1907, when the total death roll in that one year from so-called "windy shots" and powder explosions reached the appalling total of 1,148. Mr. Geo. S. Rice, who summarized the results of these disasters, said that these powder and "windy shot" explosions were in reality dust explosions. While it was true that they generally originated from a blown-out shot, it was without question the coal dust that carried the flame and death in its trail.

From the above, it will be seen that no definite conclusion had been arrived at in European countries until 1909, and in the United States not until after the disastrous year of 1907. In 1910, a station was erected at Pittsburg, and a mine also was started for the purpose of finding out if coal dust could be exploded in the absence of explosive gas. The result of the findings of both the testing station and the mine was that coal dust could be exploded, and did explode, in the absence of explosive gas. In Canada, unfortunately, there are no testing stations, and some mining engineers and numbers of miners may be found who doubt that coal dust will cause an explosion in the absence of other explosive agents.

Thus, the summary of the latest reports on explosions is fairly conclusive, and the essence of it is that there are two principal causes for explosions: explosive gas, and coal dust; and of the two, coal dust is the more insidious. The fire damp carries its own flag of warning in the shape of the cap in the safety lamp, but coal dust does not attract attention until present in large quantities. Fire damp is usually of local occurrence, and, except in notable and very exceptional

cases, it is controllable by careful manipulation of the ventilation currents. If, by mischance, a body of fire damp is ignited in the mine, the force of the explosion is terrific; but the effect is localized unless dry coal dust is present, or unless, as very rarely happens, an explosive mixture of methane and air extends throughout the mine.

Various methods have been devised for dealing with an explosion that has occurred through the temporary failure of the methods for neutralizing the explosive dust. Experiments were made in the workings of Lievin. In the path of the dust explosions short zones were prepared, in which stone dusting and watering had been carried out to the limit. A study of the results of these prepared zones led to the conclusion that a moderate length of well-watered gallery has a modifying influence on the propagation of explosions, and that the efficiency of such zones, considered as means of not only moderating, but completely suppressing, the explosion, depends upon a number of circumstances, notably the intensity of the initial explosion, the length of the arresting zone, and the degree of humidity. Briefly, a zone of 100 yards, which has been treated with a weight of water equal to four times that of the stone dust, would resist, probably, the progress of the flame of an explosion occurring within the previous 100 yards, but would not be sufficient to arrest the flame from an explosion of much greater violence.

The efficiency of arresting-zones treated with stone dust only is practically nil. Even when the initial explosion is slight, and the zone is 150 yards long, the proportion of incombustible material, to be of any use, must be in excess of seventy-five per cent. Although the watered zone is clearly superior to that treated with stone dust, the method of providing arresting zones is of little value when a violent explosion occurs. When the road is dusted only, the powerful blast of air in front of the explosion mixes its inflammable dust with that of the surrounding galleries, and carries the incombustible stone dust ahead of the flame, thus rendering it useless. When the isolating zone has been freely watered, it is true that the inflammable dust it contains will not rise, but the blast of air carries along sufficient material to propagate the flame through the watered portion of the workings. Although the evaporation caused by the passage of the flame tends to cool and

finally extinguish it, the action would have to take place for a considerable time. The arresting zones, then, are not effective when a violent explosion occurs, because, owing to the evaporation caused by the ventilating air, it would be impossible to maintain, over a sufficient length, the degree of humidity necessary for success.

From the foregoing, the reasonable conclusions are that no practicable method has been found to arrest a violent dust explosion in a dry and dusty mine. It therefore becomes necessary, instead of attempting to arrest explosions, to remove the causes, and to use preventive measures as far as reasonably practicable. Let us first look into the nature of a dust explosion. Where a source of heat, such as an incandescent wire, an electric spark, or the flame from a blown-out shot, is introduced into a cloud of coal dust, rapid combustion takes place between the particles of the coal dust and the oxygen of the air. If the quantity of heat transferred to the dust cloud is relatively small, combustion may proceed only in the immediate vicinity of the heating agent. If, on the other hand, sufficient heat is available, the temperature of the dust cloud is raised to such an extent that combustion takes place, developing more heat than can be readily conducted away; consequently the adjacent parts of the dust cloud are heated to a temperature sufficient for rapid combustion, and by its combustion it develops heat, which in turn raises the temperature of the next dust layer. This process continues, and the flame travels through the dust cloud with rising temperature and increasing velocity. When the conditions are favourable a, true dust explosion results; the velocity of the flame rapidly increases to over 2,000 ft. per second, and great pressure is generated. In coal dust explosions, pressures have been known to increase to 119 lb. per square inch after they had travelled a distance of 750 ft. from the point of origin.

There are various methods of controlling, resisting, or limiting these explosions, but the two principal ones are: (1) humidifying the air currents by water sprays, and water zones; and (2) inoculating the air currents with stone dust, and stone dust zones. These zones must be within a reasonable radius of the parts of the mine where blasting is done, and it is necessary to prevent shots from being fired in any other part of the mine, other than the working places, without

the written consent of the manager, stating the conditions, and the precautions to be taken before the shot or shots are fired.

It may be interesting at this point to look into the system of stone dusting in the continental collieries. In general, stone dusting is confined to main roads and airways — sometimes the inclines as well — the precautions at the face and in the gate roads being limited to increased care in preventing the risk of ignition in shot-firing. With this object in view, the blasting charges are not allowed to exceed 250 grams, or 9 oz., in each hole. The cartridge is encased in waxed paper. Shot holes are carefully cleared of dust and are stemmed up to the mouth. A plate of sheet iron, 40 by 32 in. at least, is set up a few feet from the hole, and all coal dust in the intervening space is removed. Workings are divided into districts, which are isolated from one another by means of barrages, etc. This is to limit as much as possible the propagation of the explosion, should one occur.

To summarize briefly, some of the preventive measures are: seeing that the dust from the screen on the surface does not enter the air intake to the mine; taking extra precautions with respect to safety lamps; setting up walls of non-combustible dust or water vapour between the particles of inflammable coal dust, or, if reasonably practicable, a combination of both. These walls should extend sufficiently far away from the working places, or the places where shots are fired, to prevent ignition of coal dust from the flame of a blown-out shot, a windy shot, or a shot that has been overcharged. In addition to this, water or stone dust zones should be prepared in close proximity to the places where explosives are used, for the purpose of dividing the mine into districts, isolated from one another, and thus helping to prevent the spread of an explosion, should one occur.

It should be seen that all fine dust is taken out of the mine, as far as practicable; that roads are ballasted with non-inflammable material; that mine cars are properly built to prevent dust escaping; that sprays of water are turned on to the top of the coal to prevent dust rising from the mine cars; that shot holes are properly cleaned, and are not gripping or fast; that the charge of explosive is limited to the prescribed amount as determined at the tasting stations; that all dust is

allowed to settle before shots are fired; and that flame of all kinds is kept away from heavy clouds of dust. Tamping should be done with non-inflammable material, bearing in mind that tamping with clay containing 11% of moisture gives the most efficient results, increasing the useful energy of the shot to about 93%. Moist tamping, therefore, does two things: it increases the efficiency of the shot, and also increases the safety.

It should be seen that ample time is given between shots, so as (1) to allow the heat generated by blasting to pass off; (2) to give the surrounding air time to dilute the gases generated by the explosive, and liberated from the coal; (3) to allow the dust to settle; and (4) for the purpose of inoculating the coal dust with stone dust or water vapour within a reasonable radius of the face.

Having dealt with some of the precautions to be taken with respect to blasting, it is now important to refer to the care necessary in the use of explosives, if we are to have a reasonable assurance that no ignition of gases or dust will take place from the blasting. The ignition temperature of all explosives exceeds that of all mine gases, but, fortunately, the duration of the flame of permissible explosives is much shorter than that required to ignite the gas. The duration of the flame of permissible explosives is only a fraction of a second, while it is said to take as long as ten seconds to ignite fire damp, which is the chief mine gas.

The difference between permissible and non-permissible explosives is clearly brought out in a comparison of black powder and ammonium nitrate. The ignition temperature of the latter is 1,121 deg. C., whereas that of black powder is 2,215 deg., or nearly twice as high. The duration of the flame of black powder is between 2,000 and 4,000 times as long as that of ammonium nitrate. Ammonium nitrate is the explosive used at the present time by the Dominion Coal Company.

If the above precautions, particularly with regard to limiting the charge of explosive to conform to the standards as set by the testing stations, and neutrilizing the inflammable dust, are observed, we shall have gone a long way towards ensuring safety in our mines.

## DISCUSSION.

MR. ALEX. S. MCNEIL: I think this is a subject that should interest every member of this Society. It is a subject that mining men all over the country have been studying for a number of years. I may say that I do not believe we have a perfect system of handling coal dust in the collieries yet. Formerly we did not have very much trouble with coal dust, but now we are getting down into deeper and more dangerous levels. The question of explosions in mines is one that concerns the public of this country and also the government of this country. The mining departments of the different governments must be giving it much consideration. In Cape Breton we are getting down to a very effective system by means of watering. I have every confidence in that system, provided it is carried out in a proper manner. There are very many different ideas as to which is the best way. I think that we ought to have an expression of opinion from as many members as possible, because there are some who do not favour the watering system. Such an expression of opinion from the members of this Society would be very helpful. It would be a great help in shaping our course when we are adopting systems in future. I wish to move a vote of thanks to Mr. McEachern.

MR. NEIL NICHOLSON: I have had some experience with explosions, and I quite agree with Mr. McEachern that dust is the main cause.

MR. W. HERD: Mr. McEachern gave a resume of experiments which have been made during the past 30 or 40 years leading up to the knowledge we now have that coal dust will explode without the presence of inflammable gas. The keynote of his paper was the removal of the causes of explosions. That is a very difficult thing to do, and the next best thing is to mitigate the effects of an explosion which might occur through some unfortunate combination of circumstances. One remark in the paper is open to question. Mr. McEachern stated that a 150-yard water zone is not sufficient to stop an explosion which has accumulated any force. That might or might not be the case, but why stop at 150 yards? To effectively water, it is necessary to have a thorough intimacy of the water and the dust. There are certain cases where one could not water, and then the only method would be to use stone dust. I do

not think we need any testing stations in Canada. We can make use of the United States Bureau of Mines, which has always assured us of its willingness to help in the investigation of any matters which may lead to greater safety in mines.

MR. W. H. GRAHAM: In the course of his remarks, Mr. McEachern referred to the use of stone dust, and it occurred to me that I might submit for your consideration the use of crushed blast furnace slag. It is high in lime; it is easily procurable; it has many good features; and it would seem to me to be a cheap substitute for stone. The Steel Company could deliver any reasonable quantity at a price very much below the cost of crushed stone. Referring to the question of raising the humidity, it occurred to me that possibly the moisture, to be more effective, might be introduced in the form of steam, for it would seem that the introduction of cold water sprays would have a tendency to lower the humidity rather than to raise it.

MR. D. H. MCLEAN: I promised Mr. McEachern that I was going to criticize him, but I feel that there is not very much left to criticize. Sometimes, when I meet Mr. McEachern in our work, I feel that he is a nuisance. He is, of course, the chief inspector for the British Empire Steel Corporation, and he has got a hobby for this safety business, and it is dust here and timber there, and look out for this or look out for that. Mr. Herd just passed a remark about the number of yards of the length of the safety or stone dust zone. I can assure Mr. Herd that Mr. McEachern always puts himself on the safe side. It is the same way with all the other recommendations that he makes to us in our daily work. The only thing that I am sorry for in this paper is that Mr. McEachern did not leave some loop-hole where we could get at him for a few minutes.

MR. J. J. MCDOUGALL: I would like to ask Mr. McEachern a few questions on the conditions under which he makes his stone dust. He was very careful in avoiding any discussion along that line. My friend, Mr. Graham, was anxious to introduce the question of using slag, but I am afraid if he did so he would give Mr. McEachern considerable trouble. I must agree with Mr. McEachern that both stone dust and water spraying are necessary.

MR. ANGUS FERGUSON: One thing that attracted my attention in connection with the paper is the fact that coal dust is such a great factor in an explosion. There is, however, another factor, and I do not know whether it gets sufficient consideration or not. That is the human element.

MR. JOHN C. NICHOLSON: Probably the greatest mine explosion in this country was that in Springhill. It was considered not to be a coal dust explosion, but to have been caused by a blown-out shot. On the day preceding the explosion, the shot firer refused to fire a shot, and somebody else was in his place when the shot was fired. One hundred and twenty-five men lost their lives. We have not done any shooting in Springhill for 20 years. The coal is easy to work, and shooting is not necessary. We have carried out a system of water sprinkling for a number of years. Stone dust, while it may be effective in a good many cases, still has some disadvantages. It has been experimented with in the United States to quite an extent. Different kinds of stone dust were tried. Sprinkling, to be effective, must take care of the roof and the sides, as well as the bottom. I believe the wet zone is the most effective method of destroying the effect of an explosion, that is if it is made long enough and often enough. The explosion would either die out or would be lessened, so that the result would not be so disastrous. I happened to be present at an explosion at Springhill, and while there was little damage done underground, the force was such as one could hardly imagine. A safety lamp was thrown up on the bank, while its handle was left hanging on the nail. This was down a slope 700 feet from the surface. One can imagine what the force would be if it had three or four thousand feet to travel over, gathering, all the way, dust to intensify it. Too much stress cannot be laid on the sprinkling system. I know at the present time in the collieries of Cape Breton a great deal of attention is being paid to sprinkling. I think it is a well established fact that the explosions in this country have been very few considering the number of years the mines have been working.

HON. ROBERT DRUMMOND: I think another party and I are the only two in Nova Scotia who know of an explosion that should not have occurred, and that it was due to the old custom that when a strike occurred and the strike was settled, the manager and the men met to have a convivial

time at night. I have the report of that explosion in my possession, and only four men have ever read it: the inspector of mines, Richard Brown, on the other side of the water. myself, and a manager I showed it to in order that he should not go through the same experience. Has there ever been another case in the world where the bodies of 49 or 50 men, who were killed in an explosion, remained in the pit, as occurred at the Foord mine disaster in Pictou county?

MR. JOHN MOFFATT: We have got far past the stage when we believed that coal dust would not cause an explosion. I happened to be in Springhill when the explosion occurred. You will often hear it said that a woman prophesied that the explosion was going to occur. I was working in the mine before the explosion. I asked the assistant manager if he would release me, and I kept at him because I had worked in the Old Country districts where heavy explosions had occurred. One of the men who was working with me was left there and he died. The real cause was dust. You could not take two steps without stopping, because the light dust would rise and choke you. There is no doubt in my mind that coal dust was the cause of that explosion. To-day we are discussing not so much the cause of these things as how to prevent them. Several remedies have been mentioned, among them water and stone dust. These are the two principal remedies. Mr. McEachern has opened up a big subject which should interest the whole province, and I am sure that we are very much indebted to him. His paper was not very long: like the latest style of ladies' dresses, it was just long enough to cover the subject but short enough to be attractive. It is a paper that is very interesting to a mining student; there is much in it that is vital to us. No doubt every one of us will leave this meeting to-day feeling more optimistic regarding the prevention of explosions.

MR. D. G. MCDONALD: Once upon a time it was not easy to convince me that there was very much to this bugaboo about coal dust explosions. In the year 1911, however, I visited the Bureau of Mines at Pittsburg, Pa. We saw there a test made of a coal dust explosion; a mixture of gas and coal dust; and also a gas explosion. I can assure you I was impressed beyond the shadow of a doubt with the fact that coal dust is perhaps the greatest of all factors in mine explosions. I would like to say that, after 35 years' experience, I believe

that, with a sufficient quantity of air circulating in a mine at a proper velocity, we would not need to worry about coal dust.

MR. ALEX. MCDONALD: There is one thing I would like to say. We can use all the stone dust, watered zones, and everything else, but to prevent explosions we will have to go a little further even than that. We must see that explosives are handled properly — not only as to tamping, but more particularly as to the quantity of powder used, its placing in the holes, and so on. We have, I am afraid, a false idea about safety explosives. We can make them as unsafe as black powder by the abuse of them. Mr. Graham referred to slag being used. There is a question in my mind about this. I think possibly slag would not be good because of certain ingredients. It might be injurious to the workmen's lungs, and perhaps to the horses' lungs also.

MR. DUNCAN S. MORRISON: Personally, I think sprinkling is the best system, but I have also had some experience with stone dusting. An experiment to show that stone dust would prevent explosions was only partially successful. The reason given was that the dust was merely shale from the roof, containing in itself a quantity of inflammable matter. This would seem to suggest that, if stone dust is used, it must be stone that does not contain any inflammable matter. Arenaceous shale would be very good. Ordinary sand might be as good as anything. In the case of explosions you have to use the best judgment at your command at the time. Conditions govern and alter cases.

# MEN AND METHODS OF THE EARLY DAYS OF MINING IN CAPE BRETON.

By C. M. Odell.

Annual Meeting, Mining Society of Nova Scotia, Sydney, May, 1922

At a recent meeting of the executive of the Society it was suggested by our President that the names of our pioneer mining men, and the methods they employed in opening up the mines of the Province, were being lost sight of, and that in justice to these men an effort should be made to trace their history and place it in some form on the records of our Society. Certain is it that we owe a debt to these men of a generation or more agone, who worked under handicaps which might well have discouraged anyone; and if we expend some effort in an attempt to place their names, and the works they accomplished, on file in the archives of our Society, we are to some extent liquidating our debt and furnishing future members with a history which may stimulate them to increased effort when we in turn shall have passed off the stage. A resolution was passed covering the President's suggestion, and the next business was to find a victim. On counting noses, I was IT.

When I accepted the duty of preparing a short history of the men employed, and of the methods pursued, during the past half century of mining in Nova Scotia, I little realized the magnitude of the task I had undertaken. A slight revision of notes hastily compiled soon convinced me that the subject was too vast to be compassed in one paper, and I decided to confine my paper to Cape Breton Island alone. In this task I have received material assistance from Mr. J. T. Brown personally, and also from a free use of his splendid library. Even with this help, I feel there must be omissions of much important data, which I trust will be supplied by members present. As I hope this will be followed by a similar paper covering the same subject on the Mainland, we may call this Chapter One.

We are all fairly well acquainted with the iniquitous lease by which all of the minerals in Nova Scotia were vested in the Duke of York, and by him conveyed to a firm of London jewellers and money lenders known as Rundell, Bridge, and Rundell. This firm in turn formed the company known as the General Mining Association, which exercised their rights under the lease until 1857.

The first names to which we must uncover are those of Gessner of Cumberland county, Fraser of Pictou, and James MacKeen of Plaster Cove, now known as Port Hastings. These three men, in season and out, fought the lease monopoly with an energy which knew no stay, until their efforts were rewarded by its cancellation and the throwing open of the mineral reserves of the Province to private enterprise.

This was in 1857, at a time when the Province was devoid of capital and but little known to the outside world. The discovery of gold in paying quantities, in the counties of Guysboro and Halifax, brought the Province into the limelight, and during the next six years there was a stampede to the gold regions. During this period the great Republic became involved in a fratricidal war, and an artificial demand for coal was created. The stories of coal in Cape Breton, described by Nicholas Denys and other early writers as consisting of solid mountains of coal where neither pumping nor ventilation was necessary, were in many respects equal to the stories of the Texas oil fields, with which our mails are now being daily burdened. As a consequence, there was a great influx of American capital and American engineers. Coal mines were opened indiscriminately wherever a possible shipping port could be found or created around our coasts.

The first mention of coal mining in Cape Breton by the Commissioner of Mines is in his report for 1864. In this report he expressed regret that no inspector of coal mines had as yet been appointed, and he was unable to give a detailed report of mining operations. He does, however, mention that there was a decrease in production of coal, as compared with previous years, on account of a strike amongst the workmen: so it appears that, even at that early date, operations were handicapped much as they are at the present day.

At the southern extremity of the coal basin, the Tracey mines, opened at False Bay beach in Mira bay, were being worked at this time. The outcrop of this seam, the lowest of all the series, enters tide water at this point, dipping inland. The idea of the promoters was to cut a canal through the beach and use the small land-locked pond, or Barrasois, as a shipping port. To this end the Federal Government was appealed to, and they did make a complete survey and plans of the proposed work, which is a habit they have, just before a general election. The promoters, Messrs. Tracey and McLeod, set about developing their mine and built a small shipping pier out into Mira bay, from which they carried on shipping for some years. Their little pier withstood the ravages of the elements until the famous August gale of 1873, when it disappeared beneath the waves. Two years later, Mr. Price Prothero commenced operations in these areas, and succeeded in opening a channel across the beach, of sufficient depth to raft logs through at high tide. Most of the timber used in the original breakwater at Morien was transported in this way. This effort proved a failure and the property having fallen into the hands of some English investors some twenty-five years later, a gentleman named Hill-Catherine took charge in 1901 and, with much beating of drums, undertook to open the Mountain-Kitty mine. Some buildings were erected, a large amount of talk expended, and the whole project abandoned. The only persons to benefit by this last named undertaking were some pirates from Scatterie and nearby ports, who secured a large lot of panel doors, flooring and sheathing from the abandoned buildings. In the year 1919 the property was acquired by Mr. Vincent McFadden and his associates, and a systematic effort to open up the inner harbour was made. A powerful steam dredge, with steam pile-driver, commenced work on the harbour, the mines were pumped and opened up, and some coal was actually shipped in 1920. Then all operations suddenly ceased, and the most southerly of all mines in Cape Breton county was again in the discard.

Following along the shore from this most southerly mine, we come next to an opening commonly known as the Baird mine. This mine, like the Tracey, was first opened by a tunnel driven into the coal, where it showed in the face of the cliff,

MacDonald sank a shaft near the shore at Port Morien, erected a coal pocket some distance out in the bay, and connected it by aerial transmission line with his bankhead. His company he reorganized from time to time under various titles, *viz:* Gowrie and Block House Collieries, and The North Atlantic Collieries. The originator of these several companies was subjected to a lot of adverse criticism by the 'Baa-man' of the near-by districts, but, like all Englishmen, he kept on, unmindful of jeer or scoff. While the output of the mine was never large and the venture never promising, the promotor held on until, at a favourable opportunity, he sold out and has since lived in retirement on the results of his foresight and the proceeds of his efforts. His critics are all still dodging the sheriff or looking for a job. It is well for some of us that we always have our laugh first, as otherwise we would never smile. The purchasers of the property reorganized under the name of Boston and Morien Coal Company, and after a few years of patient effort with no goal in sight, they backed off the stage and the property was sold, lock, stock, and barrel, to the Dominion Coal Company. Adjoining this property to the north is the site of the Block House collieries, at one time the scene of greatest activity on the Island.

Block House colliery, as its name implies, was the site of an old French block house erected during the French occupation of Louisburg in 1725. The garrison and citizens of the town were supplied with fuel from this source. The block house was for the protection of the mine workers, as there was always a scrap on with someone — and we don't seem to have quite eliminated the habit yet.

These areas were secured, and the mine opened, in 1859 by Marshall Bourinot. Mr. Bourinot worked the mine until 1864 when he sold it to a New York company composed of three brothers, Robert, Charles, and Augustus Belloni, and their brother-in-law, Charles Havemyer. These gentlemen organized the Block House Mining Company and proceeded to develop the mine on a large scale, for that period, with Mr. Crandall, of Marine Slip fame, as manager. A shipping pier was built close to the slope mouth, but as there was practically no protection from storms from the Atlantic, its upkeep was a costly item. The usual charges of extravagance were made

by onlookers, while on the other hand it is claimed by many that coal was delivered from the mine tubs to the vessel's hold at a cost of 60 cents per ton, and that, during the last year of the American Civil War, it was sold in New York at $9.00. At this date, the output of the mine was 450 tons per day. No company could stand such prosperity as this, and, as a natural consequence, the company went into bankruptcy in 1872. New capital was raised, however, and the company was reorganized under the name of the Block House Coal and Railway Company. Mr. Belloni at a later date became interested in the International mines at Bridgeport, and he had a railway survey laid down from the International mines to Louisburg, following the shore as far as Mira bay, with the idea of furnishing transport for the product of the many small mines then in operation around the shore. Mr. Belloni also employed two eminent mining engineers, Mr. Benjamin Smith-Lyman and Prof. Lesley, in connection with his projects. These gentlemen made extensive surveys and have left us a wealth of information concerning the geology and surface conditions between Lingan Basin and Port Morien.

Misfortune seemed to dog the footsteps of these energetic Americans. Their shipping pier was destroyed by the August gale of 1873, and in 1877 they were again in difficulties, when they once more shifted base and were re-organized as the Block House Coal Company, under which title they continued in operation until 1885 when they quietly passed off the scene. Their areas and all their assets were subsequently sold by the Local Government of Nova Scotia to satisfy claims for overdue royalties, and eventually the property came into possession of the Dominion Coal Company. As these areas embraced a large tract of the Gowrie seam, which is still unworked, there is no doubt that the site of old Block House will again be the scene of great activity at no distant date.

In 1863 a company composed of Messrs. Ross, Kaye, and Symonds commenced work on a slope at Schooner Pond, which they named the Acadia Mines. The Mr. Ross here mentioned was the father of the late A. C. Ross and is still familiarly referred to by the oldest residents as "Hughie" Ross — the name Hughie arising from the fact that his name actually was Hugh E. Ross. This property was on what is now known as the

Emery seam, being a continuation of the seam of that name opened the same year at Reserve. Mr. Ross, however, named it the Ross seam, and it is so referred to by Brown and Fletcher in their reports. About the same time Mr. Ross opened slopes on a seam near Low point, which he also named the Ross seam under the impression that it was the northern extremity of the same seam being worked at Schooner Pond.

There being no harbour, the coal was loaded into scows which were towed out to schooners anchored in the bay and the coal transferred to their holds. This was the only method of transportation attempted until 1874, when Reserve, Emery and Schooner Pond mines were amalgamated under one company and Schooner Pond was linked with Reserve by a line of railway which connected with a shipping pier in Sydney harbour. This company, the Glasgow and Cape Breton Company, with F. N. Gisborne as general manager, was financed by English capital and had the same ideas, and worked on the same lines, as followed later under more propitious circumstances by the Dominion Coal Company. Their Reserve areas, so called from being held in reserve by the Nova Scotia Government as a prize for a company which would undertake to build a railway from Sydney to Louisburg, were some of the richest in Cape Breton county.

In the following year, 1875, they extended their railway to Louisburg, this giving them access to open shipping all the year round and connecting with five different mines, *viz*: Reserve, Emery, Lorway, Balmoral, and Schooner Pond. Of these, the Reserve, Emery, and Schooner Pond were the only mines worked to any considerable extent. This company was in advance of its time, and was soon in financial difficulties. Sold under foreclosure in London, all its mines, piers, railways, and franchise brought but £12,000 under the auctioneer's hammer. Re-organized under the name of the Sydney and Louisburg Coal and Railway Company, it was successively managed by F. C. Kimber and D. J. Kennelly. Its railway branches to Louisburg and Schooner Pond were closed, it devoted its energies to developing Reserve mine, with Sydney alone as its shipping port, and it enjoyed an era of prosperity until 1893, when it was merged under the management of the Dominion Coal Company, which was formed during that year.

The Gisborne Company, as it was commonly called, found itself handicapped for want of a market. Indeed, at this period, other ventures of a similar nature suffered the same experience, to their sorrow. The American Civil War having passed into history, the citizens of the great Republic returned to civil life and supplied their own wants. To secure their home market to their own citizens, a city of $1.25 per ton was imposed upon bituminous coal entering the United States. This practically closed that market, and it was not until many years later that Canada, under a protective tariff, was able to absorb the product of its own mines in its entirety.

We have drifted far inland and will now return to the shore line collieries. A few miles northwest of Schooner Pond, and still along shore, we come to the Clyde, or Ontario, mine. It may be worth mentioning that *en route* we pass No. 6 mine, rendered famous by the Steel Company's lawsuit of recent date. The Clyde mine was opened by Messrs. A. and J. Campbell in 1863 near a small cove where Phalen's brook enters the sea, and here the famous Phalen seam took its name. It is rather anomalous that the counsel and experts for the plaintiff in the famous Steel-Coal difficulty spent much time and money in an effort to prove that the Phalen seam did not exist south of Big Glace Bay lake. For a few years the Clyde mine shipped from a small wharf in the cove at the mouth of Phalen's brook, but the usual failure and re-organization overtook this company like all others of this date, and it was reorganized under the name of the Ontario mine; and, by an arrangement entered into with the Caledonia Mining Company on the northern side of Big Glace Bay lake, an artificial harbour was constructed and in 1872 connection by a short line of railway established. This mine was successively worked under management of Mr. Jas. T. Burchell, John Sutherland, and Alex. McPherson. The names of J. Langdon, of Pittsburgh, as president, and Alex. Fulton, as manager, also appear in the list of prominent men engaged in this enterprise in these early years of development of our mineral resources. Having been acquired by B. F. Pearson in 1892 as the initial purchase of Cape Breton Mines, it was absorbed in the merger of 1893.

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In 1865 the local government appointed John Rutherford as Inspector of Mines for the Province. Robb and Fletcher commenced a systematic geological survey of the Island, and gave us their splendid maps and reports. W. H. Parrott and Stephen H. Head, from our southern neighbour, and Robert Drummond, from bonnie Scotland, all landed on our shores, and Prof. Howe and Henry Yule Hind lent their scientific knowledge. William Routledge and Henry Mitchell, of merrie England, came to lend a hand in exploiting our natural resources, and a season of great activity was inaugurated. Henry Poole, Wilson, Converse, Howe, Emery, Burchell, and Archbold are names which attained prominence in this immediate vicinity, while Cadegan and McLeod were busy at Bridgeport, P. Collins and A. J. Campbell at the northern extremity of the basin, and the Sydney Collieries Company were making extensive preparations to open up the famous Bonar-Head submarine areas. Hugh Ross divided his energies between preaching the gospel and opening up slopes at Victoria in this county, and at various points in Inverness. Cossitt Brothers, sons of the Rev. Ranna Cossitt, opened a small outcropping some three miles south of Sydney. William Ransome carried on boring operations between Marion Bridge and Loraine. Discoveries of coal and iron ore were reported at Glengarry and Loch Lomond, while manganese ore was opened up and mined to some extent in the Salmon river district by E. T. Moseley. Extensive works were carried on in explorations for copper at Cheticamp, there were borings for oil at Lake Ainslie, and the lead-silver mines of East Bay attracted attention. Richard Brown, manager of the G.M.A. at Sydney Mines, published his famous works in the form of a history of this Island, and a special edition dealing with the history of the coal mining industry alone. Caledonia mine, on the Phalen seam, was opened by a company composed of Messrs. Converse, Howe, and Emery, with Henry Poole as manager. After spasmodic efforts to maintain a small shipping wharf at the outlet of Glace Bay lake, this company built a railway across the sand bar, and built or constructed an artificial harbour where a depth of seventeen feet of water was maintained and shipping carried on for some years. This ambitious scheme proved unsatisfactory, and, after ten years' battling with the elements, an arrangement was made in 1884 with

the Little Glace Bay Mining Company, by which Caledonia shipped in what is now Glace Bay harbour on payment of 12½ cents per ton for the privilege. David MacKeen was now manager, and practically owner, of Caledonia Mines. This arrangement continued until the absorption of Caledonia by the merger of 1893, when Mr. MacKeen became the first manager of the Dominion Coal Company. Henry Poole had resigned his position as manager in 1872, and had succeeded John Rutherford as Inspector for the Province. In his first report as Inspector, he calls attention to the absolute necessity of having certified managers and special rules governing the operation of mines, and, acting on his advice, the Mines Regulation Act of 1874 came into force. Mining was now taking actual shape in our Province. The next in order are the works at Little Glace Bay. E. P. Archbold having acquired areas in this vicinity, work was commenced in 1858 under management of Henry Mitchell. A small wharf was built at which scows were loaded, and from these the coal was transferred to schooners lying in the offing. The old French workings of a century and a quarter earlier were re-opened at Burnt Head, in what is now known as the Hub seam, and a small shaft, known as the Harbour pit (from which the Harbour seam takes its name), was opened near the present bridge on Commercial street, Glace Bay.

Great differences of opinion as to when the weather was fit for loading occurred between the manager and the mutinous crews of the scows — the manager's idea being to get the scows cleaned for re-loading, while the idea of the crew was to fill in time and draw a day's pay. Some of their descendants are still with us. Stories are told of pitched battles between the land forces on the wharf and the naval forces afloat, in which the output of the mine figured as ammunition to such an extent that it was at last decided that this was too expensive a means of transportation, and the company known as Little Glace Bay Mining Company was incorporated in 1861. Sinking of the Hub shaft on the Hub seam was commenced, the Rooste houses were erected, Sterling Shaft was started, and dredging of the Glace Bay harbour on a well-defined plan was commenced. Jetties at the mouth were built out to a depth of 18 feet of water. Railway lines to connect the new shafts of the Hub

and Stirling with the harbour were built and a veritable hive of industry was opened up in the Glace Bay district. Mr. Charles Rigby was the manager of the company when the property was absorbed by the Dominion Coal Company in 1893.

This same year, 1861, saw the commencement of mining near the present site of the town of Bridgeport under a company incorporated under the name of the Union Mines. This company was the successor of Messrs. McLeod and Cadegan, who had opened slopes and shipped coal from a small wharf in Deadman's cove for some few years previously. As they never seemed to agree between themselves, they employed Mr. George E. Burchell as partner and general manager of the venture, and shipping was successfully carried on under his direction for a number of years.

In 1864 this company, under the presidency of Gen. Strong, with A. C. Morton as chief engineer, commenced the survey of a railway line from the pit mouth to Sydney; James T. Burchell was employed in a junior capacity in this survey. The usual fate overtook this venture, and in 1869 it was re-organized under the presidency of A. C. Morton, under whose regime the sinking of the International shaft and the construction of the railway to Sydney were commenced. William Routledge, who had been manager of Lingan mines since the year 1865, was employed as manager, and, in commemoration of the turning of the first sod of the shaft and the railway, he gave a luncheon at the MacKenzie House in Sydney to his most intimate acquaintances in the mining game. It may be of some interest to give the names of his guests on that occasion. They were as follows: A. C. Morton, President, International Coal & Railway Company; Duncan McDonald, Contractor, Coal & Railway Company; Henry Poole, Manager, Caledonia Mines; William Routledge, Manager, Union Mines; George E. Burchell, Manager, International Mines; J. P. Lawson, Manager, Victoria Mines; Alfred MacKay, Secretary, International Mines; A. M. Cody, Engineer, International Mines; J. W. Jennings, Engineer, International Mines; James T. Burchell, Engineer, International Mines; David MacKeen, Collector of Port Caledonia. With the exception of Mr. James T. Burchell, these gentlemen have all passed away.

The railway was completed and now forms part of the Sydney and Louisburg Railway. It appears, however, that this company was like the proverbial young bear and its troubles were all ahead of it. In 1877 the property was seized by the sheriff at the instance of Duncan McDonald, the contractor of the railway. For a time it was worked by his nephew, known as Clan Ranald McDonald, and was soon after leased by Robert Belloni, who employed John Johnstone as manager, with Patrick Neville as underground manager and Alex. McEachern as overman. It was during Mr. Belloni's lease that he conceived the idea of connecting all the mines then in operation with Louisburg harbour by rail, and to that end he had a railway survey made from International Mines to Louisburg, following the shore for the greater distance. Eventually this property came into the hands of Hugh McLennan, of Montreal, through a judgment obtained by the Banque Jacques Cartier. Mr. McLennan's son, the present Senator John S. McLennan, assumed managership of the mine and conducted it with much success until it was sold to the Dominion Coal Company, of which John S. McLennan was one of the original promotors and its first treasurer. Some years before the actual flotation of the present Dominion Coal Company, Mr. McLennan propounded a scheme in line with the present system of operation, and, with that end in view, he employed a promising young engineer named P. L. Naismith to locate an extension of the International Railway to Louisburg. Mr. Naismith located a line which was practically that followed by the engineers of the present Sydney and Louisburg Railway. He continued in the employ of the Dominion Coal Company after the consummation of the merger, having charge of operation of railway and shipping, and showed himself to be a man far above the average in ability. His talents, however, were not appreciated in this vicinity, and he emigrated to Western Canada where he attained fame and fortune, and is still enjoying a reputation as one of those who have contributed one big man's share towards the making of Canada.

Further along shore, near the south end of Lingan bar, a slope had been opened and a shaft sunk by the General Mining Association many years prior to this. These openings

were connected with a shipping pier on the south side of Lingan Gut by a line of railway across the beach. The community was known as "down along" — being "down along" shore — for some time after its founding, but was eventually named Bridgeport after Robert Bridge, the cashier of the company and a lineal descendent of the original firm of Rundell, Bridge, and Rundell. These operations were carried on under the management of George E. Burchell, but were closed down many years before the cancellation of the G.M.A. lease. After lying dormant for a number of years, the Bridgeport shaft and area were first leased, and finally sold, to Henry Mitchell, by whom mining was carried on with much success until the absorption of the property by the Dominion Coal Company in 1893.

The Gardiner mines, still farther west, were opened by the Gardiner Mining Company of Montreal, with William Routledge as manager. Mr. Routledge is a mining engineer whose works and writings have contributed greatly to our knowledge of mines and minerals in Cape Breton. He was successively manager of Lingan, International, Gardiner, and Reserve mines. He was a member of the Provincial Mine Examining Board, and died suddenly in 1895 while on his way to attend a meeting of that Board.

The Gardiner mines continued in operation until 1871, when all work ceased. Taken over and re-opened some thirteen years later by Mr. James T. Burchell, the property was sold to the Dominion Coal Company in 1893, and closed soon after. This mine was worked on a seam which has been proved as the lowest of the workable seams in the Glace Bay basin, and it is the most westerly of the openings yet attempted. This practically exhausts the Morien and Glace Bay basins, and brings us to the Low Point – Lingan areas, now known as the New Waterford areas.

For some years prior to its acquisition by the Dominion Coal Company, the Low Point – Lingan district was worked by the Low Point, Barrasois, and Lingan Mining Company, a subsidiary of the General Mining Association, with T. J. Brown as manager. At the extreme southern end of this basin, the Lingan slopes were opened in the early days, and a mere prospecting outfit was maintained there until 1865,

when William Routledge was appointed manager of Lingan. Under his management new slopes were opened, a railway from pit mouth to a safe shipping port was built, and a mining plant installed. At about the same time the opening of the Barrasois seam at Barrasois pond was commenced. A slope was opened on the north side of the pond, and the haulage engine erected on the south side. The two were connected by a trestle across the upper end of the pond. The idea was to dredge the pond to a depth sufficient to float schooners, which were to be laid up alongside of the trestle. Here they were to be loaded direct from the mine tubs, as they came from the slope mouth. This proved to be another bright idea, which drifted out on the fogs off the coast and was lost in the blue haze.

William Routledge was succeeded by Donald Lynk as manager of Lingan. In 1883 the famous Lingan strike occurred, and shortly afterwards the Lingan pits were allowed to fill and the G.M.A. devoted its entire energies in this basin to the new Victoria slopes. As has been mentioned already, these had been first opened in 1865 by the Rev. Hugh Ross, and the seam had been named by him the Ross seam. This opening of Victoria slopes by Mr. Ross was the first attempt at sub-marine mining on the Island.

In 1870 the Victoria Mines Railway was opened to South Bar, and shipments commenced in due form. They continued until 1882 when New Victoria was opened on a larger scale and all modern improvements, such as strike committees and P.W.A. resolutions, were introduced. The operations were carried on under Donald Lynk's management and a lot of exploratory work between Lingan and Low Point was done at the same time under the direction of Donald Lynk, William McNamara (father of John McNamara), and the late Patrick Neville.

Rumours at that date were rife concerning the location of the world-famous Mullins seam, but its position was never fixed until, by special request, the author of this epistle took the matter up and, in an article to the local press, settled once and for ever this much mooted question. At least he thought he did, but recent press notices credit this seam with a new resting place between Mullins farm and South Bar.

The Crandall seam and the Paint seam were located and named by Crandall and Henry N. Paint who were actively engaged in coal mining in this vicinity during these years. Victoria mines enjoyed a long season of prosperity under the successive management of Donald Lynk, Joseph Hudson, Robert Robson, and T. J. Brown, until 1899, when it was closed by the then management of the Dominion Coal Company. Since that date the abandoned railway has been handed over to the county authorities for use as an auto road, the whole peninsula between Sydney harbour and Lingan basin has been opened up, a new railway connection with International pier in Sydney harbour has been built, and the town of New Waterford has sprung out. As the whole of this areafrom Sydney harbour to Mira bay is now practically under control of this great corporation, a word in connection with the birth of the latter may be in order.

In the year 1893 the formation took place of the Dominion Coal Company, by which practically all the coal areas south of Sydney harbour were consolidated under one management. Various claimants aspire to the honour of having started or suggested the formation of the company. F. N. Gisborne, D. J. Kennelly, and the Hon. J. S. McLennan each had the idea. As late as 1892, A. C. Ross and his associates were incorporated under the Cape Breton Coal Shipping Company with an amalgamation idea very similar to that which was consummated a year later, by what was known as the Whitney syndicate. As a matter of fact, business conditions were such that the coal trade could no longer be carried on under the old regime. With our shores strewn with the wrecks of a dozen small shipping wharves, with competition so keen between a score of small rival concerns that prices would no longer pay running expenses, with each new influx of capital swallowed up in repairs and maintenance, with valuable properties sold and resold by the sheriff for a mere fraction of cost, some system of re-organization was inevitable. It was *self-preservation* that first suggested the formation of the Dominion Coal Company. All stories of mixing of mail matter through similarity of names, about "fortuitous circumstances", and other hot-air anecdotes, belong to the realm of romance. The mines in operation which were absorbed by the Dominion Coal Company were as follows:

| Colliery | Company | Manager |
|---|---|---|
| Bridgeport | Henry Mitchell | Henry Mitchell |
| Caledonia | Caledonia Coal & Railway Co. | David McKeen |
| Gardiner | John E. and Jas. T. Burchell | Jas. T. Burchell |
| Glace Bay | Glace Bay Mining Company | Chas. Rigby |
| Gowrie | Archibald and Company | Chas. Archibald |
| International | International Coal & Railway Company | Jos. Hudson |
| Ontario | Halifax Coal & Iron Company | John Sutherland |
| Reserve | Sydney & Louisburg Coal & Railway Company | D. J. Kennelly |
| Victoria | Low Point, Barrasois, & Lingan Company | T. J. Brown. |

In this honour list must be included the name of Chas. Mitchell, who was associated with his father at Bridgeport and who is still happily spared to enliven us with his humorous stories of men and methods of his earlier years; while Messrs. Isaac and George Greenwell still hold the stage as honoured members of our Society.

Of these mines, the Bridgeport, Gardiner, Ontario, Glace Bay, Gowrie, and Victoria were soon afterwards closed, as it was thought that sufficient coal for all markets obtainable could be produced from the remainder.

As already mentioned, the late Mr. David MacKeen, afterwards Lieut. Governor of the Province, was the first general manager of the company, and Senator John S. McLennan was treasurer. The head office was in Boston. In 1901 Canadian interests obtained control, and, with James Ross as president, a new system of management was installed under the general managership of Cornelius Shields, who in May, 1901, succeeded Hiram Donkin. T. J. Brown, who at this date was mine superintendent of the Dominion Coal Company, resigned to assume the position of general manager of Scotia Mines. He was succeeded by an American mining engineer named Karl Ludwig. Mr. Shields resigned in September, 1902, and was succeeded by G. H. Duggan, who in turn was succeeded by D. H. McDougall, the present vice-president and general manager of the British Empire Steel Corporation. H. J. McCann fills the position of manager of the Coal Company at present date.

Prominent among the many names in the official directory of this company are those of A. J. Tonge and Walter Herd, two English mining engineers of wide repute; John Johnstone, Alex. McEachern, M. A. McInnis, and Alex. McDonald, native-born Cape Bretonians, who have filled positions as

superintendents of the various districts; F. W. Gray, whose writings furnish an imperishable monument to his genius; and our Society's vice-president, Alex. S. McNeil.

On the northern side of the harbour, the main works have been carried on without interruption since the days of the General Mining Association, under the successive management of Richard Brown, his son R. H. Brown, and Thomas J. Brown. At present this district is controlled by the British Empire Steel Corporation. Many small ventures by private individuals have been born, and have bloomed and faded with the regularity of the flowers that bloom in the spring. Among these may be mentioned the Greener mine, opened originally by John Greener, an English mining engineer, who first appeared as manager of Vale colliery in Pictou county. This mine was later worked by the North Sydney Mining and Transportation Company, and is at present operated by the Indian Cove Coal Company. The latter company now operates the Greener pit, on the shore of Sydney harbour, and the Tom Pit, where the Indian Cove seam crosses the Canadian National Railway. Others of these small mines are the MacKay, but recently worked by William A. MacKay; and the Collins. The last-named was originally opened by Patrick Collins, by whom it was sold to Messrs. Guthro and Laffin, and by them to George L. Dix; then followed re-organization under the Toronto Mining Company, and work was resumed with George Scott as manager; but shortly afterwards operations ceased and the property remained idle for many years until re-opened by W. A. MacKay, who in turn passed it over to the Bras d'Or Mining Company.

The New Campbellton mine, originally opened in 1862 by the Hon. C. J. Campbell, passed through many vicissitudes, having lain dormant at one period for many years until re-opened by Messrs. Burchell in 1894, when for a short time this company shipped both coal and dolomite from the same pier. At present the property is again active under the management of the Anglo Coal Company, of which John C. Douglas, ex-M.P., is chief officer.

The Ingraham mine, opened on the northern side of Little Bras d'Or, the Matheson, Black Rock, Stubbert, and Lloyd's Cove, are a few more openings over which the Atlantic

waves sing a requiem. During these same years much effort was spent in prospecting for iron ore at East Bay and Loch Lomond by E. T. Moseley, while George's River district was explored by Mr. Ingraham. Rumours of coal in Glengarry district found many followers. Coal actually was found there in 1863, but although well-nigh sixty years have since passed, no seam of sufficient definiteness to induce the necessary capital for working it has yet been located. Much labour of an exploratory nature has been expended in Glengarry areas by Messrs. Wilson Lawlor and John Barrington, of North Sydney, and many boreholes have been sunk in the Salmon River and Mira districts by Cotterell and others, but without other than negative results in these last named localities — Glengarry, however, still has many enthusiastic champions and may become a producer of commercial importance.

In Richmond county, Sea Coal mine at River Inhabitants Basin, and a seam at Little river some three miles inland, were opened up at considerable cost. The former has been worked spasmodically for the past thirty years and is to-day being exploited. The latter was opened in 1865 by a shaft some sixty feet in depth, which was connected with tide water by a railway three miles in length. For some reason this shaft was never used, and the company was still-born.

On one occasion the Springhill Company, of Cumberland county, became obsessed with the idea of opening a coal mine in Cape Breton. Here was a mine all ready and very much for sale. One Alexander Dick, our own Sandy, was then a young man, with exalted ideas of his ability as a mining engineer; and during a fit of temporary aberration, this company employed Sandy to inspect and report on this mine. Alexander arrived at Port Hawkesbury arrayed in knickerbockers, with regulation stockings, large buckle low shoes, deer-stalker hat, and all else *ad lib*. Suspended in festoons around him were the usual small leather cases containing compass, barometer, thermometer, micrometer, and every other ometer known to science. An old fashioned rig was there to meet him, and Sandy, being young at the game, commenced to take chips off the old fashioned driver. On the way to the prospect he criticised the horse, the harness, and even the beard of the Prophet who was acting as Jehu; but Nemesis was waiting

just around the corner. Disdaining a suit of oilskins, Alexander divested himself of all superfluous gear and descended by a temporary tub and windlass which had been installed for his reception. Sandy was landed at the bottom of the pit, which he had been assured was quite dry, only to find some two feet of pit water awaiting his reception. The remarks which came up the shaft gave his hearers the impression that he had struck a twin gas feeder. On his return to the surface he expressed himself volubly concerning the mine, the water in it, the owners (of whom three were present), and everything else within range; when one of them interjected, "Then you don't think you'll buy the mine?" Sandy's reply, which is unprintable, convinced them that, in his then frame of mind, such was not his intention; whereupon his interlocutor remarked "Then you walk hom', you son ————", and, suiting action to the word, the three of them bundled hastily into the old wagon and, putting the gad to their aged equine, they left dripping-wet Alexander, festooned with all his mining gear, a long seven miles from home. Since that date Mr. Dick has carefuly refrained from expressing an opinion of prospective purchases until he is safely ensconced in his office chair.

Inverness county has also offered inducement to the investor and prospector. A seam of coal at Port Hood was opened in 1865 and for some years spasmodic efforts were made to establish a mining industry. Reports at that date refer to the safe harbour nearby. This was true at that date, as a bar at the northern end of Smith's island connected the island with the mainlaind and formed a perfectly safe harbour, until the fishermen opened a channel through the bar in order to bring their fishing boats to a safe anchorage without going around the island. Unfortunately, the elements finished the work and washed the bar completely away, so that but little protection is offered at present. In 1900 this mine was again opened under the management of John Johnstone, and it was steadily worked by Mr. Johnstone and his successors until 1911, when a feeder of salt water broke through and completely flooded the workings. Re-opened on a small scale by Malcolm Beaton and his associates in 1918, it was again about to be closed in 1921 when the United Mine Workers undertook to keep it in operation, and actually did work it

for a few weeks from August 12th until Sept. 16th, when they abandoned their maiden effort at mine management and the mine again reverted to its original owners, with its future beclouded by mechanics leins, judgments, injunctions, and various other legal handicaps.

Twelve miles north by rail from Port Hood is the village of Mabou, and down shore some five miles is the site of Mabou mine. This mine was opened in the early sixties by Hugh E. Ross and others. Lack of transportation facilities of any kind caused its early abandonment. In 1893 the Mabou Coal and Gypsum Company opened up some gypsum quarries in the vicinity and made a feeble effort to work the coal mines in conjunction therewith. Their methods of working would be laughable were it not for the serious side presented to the investors. Eventually the shareholders took action and landed more than one of the chief officers in durance vile as a result of a conviction for fraud and mis-management.

In 1901 Mabou was again opened by a company which seemed to be possessed of unlimited capital. They built houses, installed all plant and machinery, erected a shipping pier in Mabou harbour, and connected it by rail with their works. They continued in active operation until 1908 when someone pulled the string and the curtain fell. The local government took hold, and through its Mines Department undertook to keep the mine pumped out. This they did from Sept. 4th, 1908, to January 17th, 1909, when a serious leakage in the workings near the shore developed. For some reason, the officials of the Mines Department made no effort to stop this flow, and in seven day's time the whole mine was completely flooded and has remained so ever since.

Along shore another seventeen miles, and we reach Inverness, where collieries were first opened in 1865 by Blanchard and McCully, under the name of Broad Cove Mines. The usual scow and schooner method of shipment was introduced, and the usual failure followed. After some years of patient endeavour, the mine was abandoned until 1893, when a company, organized by A. C. Ross under the name of the Boston and Nova Scotia Coal Company, undertook to open the mine and connect it by rail with tide water at the Strait of Canso. This company, of which the writer was chief engineer, struggled

manfully through the great financial depression of 1893-96, when, in company with many other enterprises of these fateful years, it was relegated to the bone yard, to be revived a few years later by Messrs. Mann, MacKenzie and Company, who, with Government assistance, built the railway and opened up the mine, which they have kept in operation ever since. Unfortunately it is now in the hands of the Receiver, and is being operated for the benefit of the bondholders.

Twelve miles further north, on a small basin known as St. Rose, a pit is being kept open for local consumption. The operating company at one time had visions of a rail connection with the C.N.R. at Orangedale, but their efforts only extended to the location of a route *via* Lake Ainslie.

Some four miles on and we reach Chimney Corner mines, the most northerly of the Inverness group, opened by Mr. Evans in early days. Backed by English capital and with a small sheltered cove in which a shipping pier was built, it enjoyed a season of prosperity, extending over some seven years. Then, on March 3rd, 1873, a fire destroyed all surface workings, and a few months later the August gale demolished the shipping pier, and put an end to all coal mining efforts on that side of the Island for many years. Operations have been revived within recent years under the guidance of Dr. Chisholm, M.P., of Inverness, with Mr. Evans, a son of the original manager, in charge, and the management is still cheerfully looking a payroll in the face.

The mineral products of the Island are not confined to coal alone. Copper mines at Cheticamp were extensively explored; gypsum quarries at various points were opened up; borings for oil were carried on at Lake Ainslie for many years; talc, or soapstone, was discovered at Brigend, near Whycocomagh, and alongshore at Kennington cove. Iron ore areas were taken up, and much money was spent in proving them, while gold was discovered and actually worked at Cheticamp and Middle River. Plumbago (graphite) mines at rear of Benacadie have engaged the attention of mining engineers on several occasions. N. J. Brown is credited with the discovery of marble deposits at West Bay, from which locality the Dominion Iron and Steel Company has drawn its supplies of limestone for many years. It seems to be the fate of all new

mineral discoveries to be grossly exaggerated in the first reports, and in this case we have no exception, as the original reports of this marble claim that, for building and ornamental purposes, it is comparable with the world-famous Vermont and Tennessee marbles.

In early days Caleb Huntington developed a deposit of brick-clay on the south side of Mira river, and found a market for his product in the growing town of Lingan. This brick yard has been worked intermittently ever since. A deposit of material from which splendid samples of fire brick have been made has been opened near Leitches creek. To develop this, the Scotia Fire Brick Company, with the late Graham Fraser at its head, was incorporated, and work of an extensive nature planned, when the sudden death of Mr. Fraser upset all plans of the promoters. Nearby areas have since been developed by the D. I. S. Co., and a large quantity of fire brick has been manufactured.

The East Bay silver-lead mines loom on the horizon, at intervals, but even the eloquence of E. J. Brown (another of the same name), in his descriptive articles, has failed to galvanize them into active operation. The late Hector F. McDougall, for many years Federal Representative of Cape Breton county, spent the greater part of his energetic life in prospecting the mineral wealth of his native county.

One other mine which passed the mere development stage is the Coxheath copper mine. Opened in 1863 by Jas. MacKenzie, it was brought to the notice of Isaac P. Gragg, of Boston, when that gentleman was interested in Lake Ainslie oil wells in 1881. Extensive development was carried on under successive managements for some twenty years or more, but, for what cause I know not, the actual working of the mine as a copper producer was always deferred. This undertaking enjoyed a reputation as a producer of works of fiction. One gentleman, lately returned from Arizona, was fond of relating a story of a dark night, a deep glen on the Coxheath road, a bag of dollars for the payroll, a hold-up by masked men, a lot of real-sure-enough gun play, and a rapid retreat of Robin Hood's descendants. As this was during the good old Scott Act days, when fighting whiskey was to be had for the asking, it is staggering to think what might have happen-

ed if this show had been staged on present-day 'moonshine'. Just one more story to end this dreary tale, and I will ring off. There lived a character who owned a saw mill and supplied various mines with timber. He was generally known as "Ugly Duncan". I have seen the gentleman, and can vouch for it that he earned the title. His specialty was a huge mouth. When speaking to him I was always afraid that, when he opened his mouth to speak, the top of his head would fall over backwards. Upon one occasion Duncan took a contract to supply certain timber to one of the mining companies, and finding that he was losing money, he was desirous of breaking the contract. The company was obdurate and insisted upon his living up to the terms of the agreement. Duncan questioned the terms, and asked to see the agreement. The obliging clerk produced the document, and spreading it before him, said, "There is your contract; read it for yourself". Duncan did not waste any time reading it; he simply grabbed it, stuffed it into his mouth, and swallowed it. This was before the discovery of the X-ray treatment, so the contract has not been read since.....

With but few exceptions, these men of the mining game have passed off the stage. They cheerfully spent their time and money in an effort to wrest from Nature's store-house the treasures which would add to the comfort and convenience of men. Their failures are but stepping-stones for the rearguard of the vast industrial army which is ever moving towards the light, and of which our Mining Society is no mean unit. It is not the lust of gain alone which leads men out into the world's waste spaces; it is the un-named creek, the lure of the lonely silence, the long lost trail, the call of the wild — and so long as there comes a story of fresh treasure just beyond, the race will supply the man. As one of our Western poets has put it,

*"So long as Klondyke widows trail*
*our men through devious ways,*
*Someone must stick on the lonesome crick,*
*for there's ever a him that pays."*

History does not relate the name of the lady who got the jewellery, but certain it is that King George IV (while Prince of Wales) and his brother, the Duke of York, had 'run their necks' with a firm of London jewellers to such an

extent that King George, through his said brother, handed over all the minerals in Nova Scotia to Rundell, Bridge, and Rundell, in liquidation of their claim. Out of this was evolved the General Mining Association, commonly known as the G.M.A.

For half a century they exercised their monopoly, and unfortunately, in this case, it was the inhabitants of our Island who paid. It was the selfish enforcement of their rights which acted as an incentive to the men on the outside, for "the dream of the white man ever goes out to the fight that can never be won", and the men of that day were the men who knew not how to fail. To-day we honour their memory, and a half century hence, let us hope that the men who are still carrying on will be able to look back to this day and say, "God be thanked; whatever comes after, we have lived and toiled with men".

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## DISCUSSION

MR. A. S. MCNEIL: Mr. Odell's papers have always been a great stimulant to the meetings of this Society. They are always full of humour and very deep and valuable information: indeed, they are the attraction that a great many of us have in coming here. His present paper connects our past mining history with the present. It makes men of my own age ponder a little and look back. When I think of some of the names that Mr. Odell mentions, I begin to realize I am not as young as I used to be. The paper gives us a good idea of what was being done some years ago. Whether in the morning of our mining career, or in the evening of our life, we will always be glad to hear a paper by Mr. Odell.

HON. R. DRUMMOND: I shall not attempt to spoil the effect produced by the reading of Mr. Odell's excellent paper. I would call it an encyclopædia, almost, of mining events in Cape Breton. But there are one or two important omissions. He did not give us any account of perhaps the greatest mining association that was ever formed in Cape Breton. They called

it the Cape Breton Mining Association, and it was started for the purpose of showing that they could mine coal without the assistance of the Provincial Workmen's Association. In that they failed, and went into voluntary liquidation in four or five years. He did not make any mention of a person named Charles Odell, who was the only man who could tether the Mullins seam for twenty-four hours in the same place. Then again, while stating that a certain inspector of mines was the first to recommend the granting of certificates of competency to mining officials, he omitted to show you how surroundings may affect a person by pointing out that this same inspector was the chief opponent of the granting of certificates when the Bill came before the Legislature.

MR. F. E. LUCAS: As this is not a matter of deep technical discussion, I feel free on that account to rise, and in rising I take very great pleasure in moving that a vote of thanks of this Society be extended to Mr. Odell for his excellent, instructive, and entertaining paper. I am sure that to me, as well as to those of us who for so many years have not been very intimately connected with mining, it is especially interesting from the historical view point. I am sure it is a paper that in after-years will be as highly prized as, or perhaps more highly prized than, any other proceeding of the Mining Society.

MR. JOHN MOFFATT: We have listened to such a feast of good things that we do not like to get up and spoil it. We were all anxious to hear Mr. Odell's paper. The man who inquires into the past and brings on the stage different characters, as he has done in such an instructive and interesting way, is a very worthy man. There are, at the meeting to-day, a number of those he speaks of. On looking back, it seems like a long time. First he touched on Mr. Drummond. Mr. Drummond has been in this country a great many years. Mr. Odell has talked of men I have heard of after I came to this country. I never saw them, but it was interesting to me. It was very laughable to hear him describe one of my friends, Alex. Dick. He and I used to go to Jake Johnson's school. Instead of learning lessons at school, we learned stories about the Indians in the west. It shows Mr. Odell's ability to picture things

and place them in their right positions. The Mining Society is greatly indebted to Mr. Odell. If Pictou and Cumberland counties could be written about in the same manner, we would have the beginning of a very interesting history of practical mining work in Nova Scotia. Those who come after us will read the story with a great deal of interest. It is very necessary that it should become part of the history of the mining industry of Nova Scotia. It is all right to read Mr. Brown's book. Nothing much has been added to that, but a great deal has been added this afternoon to our knowledge of the history of the practical work of mining.

HON. ROBT. DRUMMOND: Each member of this Society should be given a copy of the paper in pamphlet form.

MR. C. M. ODELL: I think we can arrange that with Mr. Mackenzie, the Secretary of the Institute. Vigourous complaint was made at the last meeting that some of the papers were never published. When we became affiliated with the Mining Institute we had the understanding that they would be published. I do not know whether we count very much or not. Prior to that, all these discussions were printed immediately after the Annual Meeting. The only thing to do is to get after Mr. Mackenzie, who is expected here to-day. I think we will introduce him to Mr. Drummond, and he will probably get what is coming to him then.

HON. E. H. ARMSTRONG: I have listened with very intense interest to this paper. I hope the Society will come to some determination by which we may have Chapter Two. It would be a very happy appendix to this very valuable work. We have too little mining literature. This paper of Mr. Odell's ought to be perpetuated in some form or other. I hope some steps will be taken by the Society to have this printed as Chapter One; and have Chapter Two cover Pictou and Cumberland counties. I assure you, Mr. Chairman, that I take a great deal of pleasure in seconding this motion. I have been simply captivated by this paper. It is a splendid history, and it ought to be continued. The Nova Scotia Historical Society in Halifax does a great deal to perpetuate historical papers, and I hope Mr. Odell will be induced to write Chapter Two, and also Chapter Three if necessary.

MR. STUART MCCAWLEY: I think that what Mr. Armstrong has said is very important. The Nova Scotia Historical Society has published a lot of valuable documents in the past, and I do not see why we have not a branch of it in Cape Breton. I know another gentleman, connected with mining, who has a paper that should be published but which has not been read yet. It is wonderful. It would make a wonderful history of Cape Breton, but he does not feel like spending the money. If we had a branch of the Nova Scotia Historical Society in Cape Breton we would get a lot more papers like Mr. Odell's, and get the Cape Breton end published. It is just as important as the Halifax end, although Cape Breton is hardly mentioned there. I think the last Cape Breton article was one by Senator McLennan, on the City of Louisburg, about 16 or 17 years ago. I hope Mr. Armstrong, when he goes back home, will tell them that it is suggested that we have a branch down here to assist in getting papers like Mr. Odell's, and others that I know of, put in such form that they will be preserved.

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